A
MANUAL OF MINING.

BASED ON THE COURSE OF LECTURES ON MINING
DELIVERED AT THE SCHOOL OF MINES
OF THE STATE OF COLORADO.

BY

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PREFACE TO THE FOURTH EDITION.

BASED originally upon lectures delivered at the Colorado State School of Mines, between 1880 and 1888, the various editions received only such additions and changes as were necessary in its use as a text-book in the mining schools of this and other countries. The Author of the MANUAL OF MINING presents in this edition a complete revision of the work, which now assumes the form of a collaboration with the authors mentioned below.

Mr. E. B. Wilson, who needs no introduction to the mining public, brings to the original work a wide range of experience and ability, an accuracy of detail and a discrimination in description, contributing highly to the value of the revision. He has devoted himself particularly to Chapter III of Part I and Chapters V, VI, VII, and VIII, of Part II. Originally planned for Metal-mining, the first issue of the book gave small place to Coal and its extraction. This new edition has of necessity been enlarged to include Coal-mining in all its phases, with full descriptions and many illustrations of modern methods and machinery. The later devices in power generation and distribution, as the steam-turbines, oil, and compressed-air engines, with such appliances as have proven themselves meritorious in economy and safety, are elaborately treated. The chap-

ters upon Electricity in its Application to the Mining Industry, prepared by Mr. Rolin W. Hutchinson, whose work on "Long-distance Transmission of Electricity" has secured recognition in this field, will prove invaluable to the student.

The Author acknowledges the courtesy of manufacturers and authors permitting the use of illustrations, tables, and examples which have been inserted herein.

MAGNUS C. IHLSENG.

NEW YORK, August 1, 1904.

PREFACE.

THIS treatise is an abbreviation of a course of lectures upon mining, delivered at the School of Mines of the State of Colorado, and is issued with the advice of its Board of Trustees, which recognizes the importance of having, within a moderate compass, the best information obtainable upon this subject. In its presentation the writer has followed what his own experience has taught him to be the natural sequence, and has endeavored to introduce such matter as sixteen years of lecturing and field work have suggested as requisite. Part I contains a brief geological review and a discussion of such points as the engineer must include in his report, i.e., the preparatory and development work, systems of mining and the plant for power, hoisting, pumping, and ventilation. Part II embraces the practice of prospecting, drilling, blasting, shafting, tunnelling, and timbering, in addition to some remarks upon the examination of mines.

The work is designed as an elementary treatise for the use of those desiring a reference-book. The complexity of the subject, its extent, and the variety of machines to be described and represented, demand an elaborate discussion that would fill several quartos. Descriptions of obsolete and expensive systems or machinery are relegated to the historical works on mining. American and foreign practice is described, and suggestions for lines of future progress are offered herein. The principles of the construction and operation of machines used in mining are

explained with a perspicuity and conciseness compatible with the field in which this publication is to be sown—among students and mining men, to whom a knowledge of the fundamenta of their work is valuable, but whose acquaintance with the theory is slight.

The wants of the latter class have been kept in mind, and the writer hopes that the manual may prove of some benefit to the intelligent reader, of whom it presupposes an elementary knowledge of the sciences and of the simple machines.

The author regrets his inability to deal with the subject of "electricity in mining" as it deserves. Two reasons account for this: insufficient data, as yet; and the large space which a satisfactory explanation of the principles would demand.

The writer would also beg leave to say that the literature of mining and its cognate branches has supplied much of the material contained herein. References could not be made for each hint obtained, but obligations are acknowledged to the authors of the publications mentioned, to which the reader is referred for further details. The information has been garnered from the best available sources and condensed. The *Engineering and Mining Journal*, the *Colliery Engineer*, and the transactions of the *American Institute of Mining Engineers* have been copiously drawn from, as also the experience of the practical men, to a long list of whom the Author is indebted for many courtesies. Finally, to the manufacturers and engineers thanks are rendered for the use of the electrotypes, which have so largely contributed to make the work attractive.

MAGNUS C. IHLSENG.

GOLDEN, COLORADO, Nov., 1891.

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MINING ENGINEERING.

CHAPTER I.

PROSPECTING.

Introductory.—The search for the useful and precious minerals has been diligently prosecuted since the early days of civilization; their discovery and application have made nations powerful exponents in the world's history. And nowhere is this fact better exemplified than in our own land, in the wonderful opening and rapid settlement of the Western mining States.

No subject is more entrancing, no occupation more exhilarating, than mining, with its wonderful kaleidoscopic changes. In early times excavations were made and mines worked only to a small depth and in easy rock, and that, too, only for substances of high intrinsic value, notwithstanding the myriads of slaves to furnish the labor. The attempts at systematic mining were few and far between; but since the advent of the steam-engine, mining has been acknowledged an important profession, requiring technical education. Competition with the whole world, brought about by the improved means of communication, the paucity of bonanzas and their rapid exhaustion, compel

a skilful utilization of all the aids to a cheap extraction of our immense wealth.

The accessibility of the mine and the vendibility of its product are the ever-ameliorating features in the mining history of nations, districts, camps, and individuals, gradually divesting mining of its risks and rendering it more and more akin to manufacturing. Each new camp, untrammelled by tradition to keep it in the rut of prejudice, displays its genius for organization and absorbs the latest devices, tried and true. Nevertheless, it must be admitted that in each camp an adequate solution of the problem involves intricate questions of environment. The economy of mining is a function of many variables, as geological stratigraphy, subterraneous uncertainties, wages, water, timber, transportation, and treatment. The constants are few. The proper relation of these it is our province herein to discuss.

Hitherto a gambling spirit has frequently controlled investments in metal mines. Speculative tendencies, not technical economies, have dominated some of our operators; their heavy aggregate outlay may have proven unprofitable, for the present, because of salted mines, attractive prospectuses, or incompetent management. It must be remembered, however, that they have contributed to the prosperity of the country, and at some later date their abandoned exploitations will be pursued to profit, when the potential investment of to-day will have been resolved into future kinetic dividends, the cost of production being continually on the decrease.

Native Metals.—The occurrence of the useful minerals in the state of native purity is rare. Still less often are they found superficially: they must be delved for. In the extraction of this subterraneous material, and its delivery to the surface, consists the art of mining. The legal definition of a mine includes such "workings as must be artificially lighted."

Gold and platinum are found native in the placer accumulations of ancient and modern river-beds, which furnish fully 75 per cent of the total output of these metals. Gold occurs in segregated veins, alloyed with telluirum, and always asso-

ciated with pyrites and titaniferous iron; also intercalated between the sheets of slate or shale, or finely disseminated in eruptive rocks. The only extensive native copper deposit is the remarkable product of the Lake Superior region, where the irregular masses are mined out of the amygdaloid trap and sandstone. Singular masses of metallic iron ore are found in several localities, but they are curiosities and casual, if not meteoric. Native silver is rare and occurs in Peru, Mexico, Norway, and in the Lake Superior copper mines.

Ores.—With these few exceptions the metals are found in chemical union with non-metallic substances, more or less completely segregated to constitute mineral. Any accumulation of mineral of good quality and in sufficient concentration to warrant the expenditure of energy for its extraction is an ore. Manifestly this is a fickle term, since it depends for its stability upon the casual conditions of the market as well as upon the mineralogical features.

The most common substance is iron, entering as it does into almost all rocks and veins. Its most frequent, and valueless, combination is with sulphur. Magnetic and specular oxide and the carbonate constitute the entire supply. These occur as irregular masses in the rocks of every geological age, or in veins mixed with other minerals, but are chiefly in the metamorphic crystalline, Archæan rocks. Zinc is obtained from calamine, franklinite, and blende, which are quite extensively distributed in the Carboniferous strata. With very few exceptions, galena is exclusively the ore of lead The carbonate and the sulphide, in the lower Silurian and Carboniferous strata, mostly occur in irregular shoots and pockets, and rarely argentiferous. In the older metamorphic rocks the galena is confined in fissure veins carrying silver and gold. The main supply of silver is from its minerals, more or less intimately associated with other ores. Similarly with them, it has a wide geological distribution, and is also found "dry" in fissures. Copper, as chalcopyrite, bornite, and cuprite, is disseminated in and along slates and sandstones, rarely above the Triassic. Many galena

veins in the metamorphic rocks change with depth to copper. Mercury comes from cinnabar, which is found in true veins and in contacts. It is not commonly encountered. Tin has a characteristic occurrence in but one form, as an oxide, and only in gash or segregated veins, or "stockwerke" of the older rocks.

Tin lodes are of the segregated type, and gold or silver bearing, pyrites and cassiterite being the common minerals.

Millerite and pyrrhotite are nickeliferous and occur in gash and segregated veins, rarely deeper than 500 feet. Rich films of genthite in talc veins often constitute a commercial supply.

Manganese ores (standard contains 44 per cent of the metal) are generally associated with limonite and occur in pockets usually embedded in clay as contacts or beds or permeating slates. Films of manganese appearing in moss-like forms on the face of rock give it the name of "landscape" rock.

Mica is generally in bedded veins, instances of contacts and true lodes being rare. They are simply and always dikes in coarse granite. Hitherto only large slabs were sought, but now the fine, clean mica has a ready sale for lubrication and other purposes.

Phosphate rocks for fertilizers, the practical value of which is determined by the amount of phosphoric acid contained, are found as beds of irregular thickness; veins or lodes transversely to the strike of the strata, or superficial deposits. Apatite occurs concretionary in a clay matrix between limestone and clay. These are more frequent in the Miocene.

Many of the metals are incidentally obtained from their mineral compounds while smelting for other metals with which they are associated.

Gangue.—The metalliferous portion of a lode comprises only a small portion of its contents. The argentiferous galena, bornite, blende, or their oxidized derivatives in grains, pockets, or streaks, more or less connected, are associated with a "gangue" of clay, quartz, fluor, calc, or heavy spar. These earthy materials sometimes are intimately mixed with the mineral, and again

lie in layers contiguous with it, or the different constituents may even manifest a ribbon-banded structure.

Definition of a Vein.—The metalliferous and earthy contents of a deposit constitute a bed or a vein, and may exist under such circumstances as to render it workable. The term *vein* is intended to describe a regular unstratified deposit in a fissure that traverses the country for a considerable distance, longitudinally and vertically. The Supreme Court has defined it as "any zone or belt of mineralized rock lying within boundaries clearly separating it from the surrounding rock." This demands a well-defined crevice of ready identification, and two solid walls to give it individuality. Its lead must be metalliferous. A vein is the filling of a pre-existing fissure. The term has lost its original significance, for formerly, the mineral system was supposed to bear a resemblance to the human circulatory system. True, the fissures have originated during periods of great dynamic movement, producing folds and fissures which are supposed to have extended deep into the earth's crust, but the main artery has yet to be located. Though argentiferous lead veins are quite persistent, no evidence exists for the dogma, so tenaciously held, that they increase in richness with depth. They may or may not become richer, or change, in constituents. Examples can be cited for either side of the argument. In folded strata the deposit inclines to be thicker at the ridges, or troughs, and thinner at the sides of the folds. But this is not generally the case in massive rocks.

Usually the vein matter is crystalline. It is commonly separated on either or both walls from the surrounding rock by a sheet of clay (called "selvage" or "gouge"), or by other quite distinct lines of demarcation. The surface of contact of the deposit with the adjacent rock is called a wall, roof, or floor, according to its relative position to the miner. Not infrequently the walls are polished surfaces ("slickensides"), due to grinding caused by the slips during nature's contortions. Sometimes portions of the vein have slid on one another, causing "false walls"; therefore the miner is advised to occasionally break

into the walls to assure himself as to the fact. On the other hand, a vein may have only one or even no wall. In the process of mineralization, the original face or faces of the fissure may have become disintegrated, and all evidences of the looked-for wall obliterated. In such cases, economic, not geologic, or legal conditions define the vein.

Fissure Veins belong to regions of metamorphic action, and are the principal repositories of the precious metals. And it is a striking fact that they are rarely found singly, rather in groups of parallel veins, often in congeries. *Stockwerke* is a term used to describe a condition of affairs in which the country rock is creviced in all directions, so that the whole mass must be mined out. Some are filled with eruptive matter, others with vein matter, still others were subsequently closed without any deposition. The mineral components are markedly dissimilar, and indicate different sources. Those filled with the same variety of mineral were doubtless produced by contemporaneous forces. Those fissures which interrupt the continuity of the older veins are called cross-courses. The manner in which the intersections occur determines their relative age. Their absolute age is not ascertained, unless in stratified rock. Drags are more common than is supposed, and should not be confused with intersections. The latter are usually richer, the former not necessarily so, at the point of juncture. Many of the older veins are broken and displaced by faults. Not only do veins "pinch and shoot," but the pay streak will vary in thickness, plunge from wall to wall, or split up into numerous feeders and ramifications, and even disappear in a thread.

Gash veins hold a subordinate position to fissures, but they are of small extent, and are usually confined to a single member of the formation in which they occur. Their habitat is unmetamorphosed sedimentary rock. They have no distinct walls or gouge, and are unreliable.

Mineral Beds.—The important sources of mineral are the metalliferous deposits which occur in the sedimentary strata, and are termed beds. While the geologists may classify them,

the group is sufficiently identified by this term for mining purposes. It includes deposits, somewhat irregular in dimensions, occurring in the transverse joints of the rocks; as cementing material to the remnants of shattered or insoluble rock; as layers conformable with the strata; as isolated impregnations of grains or bunches in porous rock; or as a metasomatic replacement of porous rock. They may be found similar to fissures in a certain formation, then as a blanket contact parallel to the stratification, to again plunge into a lower series of rocks like a fissure, or branch out into a chamber. They are more easily mined, but are less persistent in depth, than veins. Their mineral contents are very compact, seldom crystalline, and the gangue hardly distinguishable from the country rock. The mineral is more or less concentrated along certain lines called "ore-shoots," which probably constituted the channels of communication with the ultimate source. The same is also true of veins.

The Theory of Vein-filling.—One requisite condition for mineral deposition is a crevice, a porous or soluble rock conduit for the fluid from which local action has precipitated the mineral. Open cavities were not necessarily pre-existing, for a vesicular rock would allow of an easy flow to the magma, or it might be equally well secured by dissolving action on the rock and a subsequent replacement. This is independent of its geologic position. In every age are rocks which will satisfy this condition. Besides this, a long train of circumstances has preceded the vein-formation involving dynamic agencies, heat and metamorphism, and even eruptive action, as important factors. These disturbances having been often repeated through the different ages, the older rocks were more frequently shaken up. Beyond this no reason exists for the prejudice which favors certain geological formations as ore-bearing.

The geognostical relations between veins and their contents are of importance to the mining engineer, but our limited space will not admit of any discussion here. The various works on geology will supply the information as to the vagaries manifested by ore occurrences and the numerous theories held. Some

isolated examples exist under such circumstances as to suggest the same origin for the ore as for the adjoining rock formations. Many of the beds and veins have been impregnated by percolating waters, perhaps at high pressure and temperature, contemporaneously with the country rock. Their metallic contents may have been carried in solution or they may have been in a molten or a gaseous state when the way for their passage was opened. This is a matter for conjecture, as is also the ultimate source of the mineral. The evidences frequently point to their deposition as sulphides, the oxidized forms being accounted for by long-continued action of atmospheric agencies. In the presence of coal and bitumen in many lead and zinc beds is suggested a theory of cause. The "water-line" theory has served its day and is no longer tenable. The current theories have offered more or less satisfactory explanation of the genesis of some of our ore deposits. Some one theory may explain some of the capricious examples of lodes or their anomalous fillings. But when we find contiguous depositions contrasting widely in point of density; narrower parts of fissures filled by denser or richer ores; superior minerals higher up than the more volatile or lighter ones, even alternating with them, it is impossible to advance a theory that is specifically applicable to all ore occurrences. Aqueous currents have apparently conveyed the minerals from their source and by evaporation chemical decomposition or electrolytic action deposited them simultaneously with or subsequent to the formation of the cavity in which they are found. The veins we find, but not always the silver; and this inability to formulate a general law by which to locate the hidden bonanzas has led to the compounding of the numerous witcheries, and divining-rods of every conceivable form, for imposing upon the credulity of the prospector who seeks a quicker means of acquirement than is afforded by the use of the pick, shovel, and patience.

There is no particular angle of dip or bearing of trend that is universally favorable to rich veins. Rules based upon such observations are local only. The same may be said as to the

supposed "live"-ness of certain rocks to mineral. Attempts to formulate indications of "quickening" mineral by associations with general gangue matter or minerals have failed of generalization. The mineral is where you find it. The Cornishman's adage, "riding a zinc horse to fortune," has no verity in this country. Each locality has its own peculiarities of mineralization, which the careful and systematic engineer will observe and regard.

With the two classes of rocks, stratified and massive, are coexistent the two classes of mineral deposits, beds and veins. Though many occurrences are of a nature that admits of question as to classification, for mining purposes a sharp line of distinction is not sought. Legal technicalities have so confused the definitions of deposits and veins as to obliterate all semblance to the original intent of geologists and mining men. Of this, more later. At present we shall consider some rules to assist the prospector in his search for mineral. And while it must be admitted that many a find has been made through accident, the existence of the ore would be found not to be at variance with the cumulative rules of geologic science.

Accordingly, the prospector will depend upon geological data. In regions of stratified rock the matter is simple. Coal is found in three geological horizons, and the presence or absence of the rocks belonging thereto is indicative of the prospects.

The metals and their minerals are distributed, geologically and geographically, over a large extent. The zinc ores in this country occur in the Carboniferous and along the Mississippi valley. The Archæan and Silurian are most prolific of the other ores. The precious metals are chiefly found in the mountainous districts, because the phenomena attendant upon their formation were conducive to the filling of veins, and the forces which gave character to the mountain also impressed themselves upon the vein, which is exposed to view and subject to location. Without some such providential occurrences to change the monotonous topography of the preadamic surface, bedded veins of the

stratified districts would have been revealed only by boring, while those in massive rocks might never have been formed.

Surface prospecting is confined, therefore, to the seeking for an outcrop. In igneous rock the outcrop is easily found. For, unless the hill is covered with slide rock, it is indicated by a jutting ledge (if the vein matter is harder than the country rock), or by a sag (if it is decomposable). In heavy timber this may go unnoticed. At high altitudes snow in the sags calls attention to the leads.

The same is true of coal, which is located by the terraces which mark the outcrop. The trend of the terrace, relative to the topography of the hill, gives a good idea of the slope of the coal. The bench itself may give the desired information, but usually it will be found that the coal dips with the hill, when the terrace or depression deflects outward toward the bottom of the hill, and the reverse for a coal dipping inward, when the outcrop will be concaved toward its top.

Substances foreign to the rock deserve notice. Alternations in the color of the slide rock covering the hill are good indications of the presence of oxidizable minerals above. So, too, vegetation is a guide. Iron springs often accompany the outcrop of coal; the ochreous covering of the rocks and soil is noticeable near some of the anthracite seams, and is common in the semi-bituminous districts. Masses of highly oxidized matter, broken from the veins, compose what are called "blow-outs," and are common in galena regions.

If no evidences of outcrop are thus found, "booming" may disclose it. During winter or a wet season, snow or water is collected in a reservoir upon the hill, and, at a convenient time, turned loose to plough its way over the soil in its fall. Many a vein has been thus discovered without great expense.

In stratified regions the order of the geological series may be observed, and certain fossils furnish the guide. Or, if the prospector is examining new ground, he has but to look for mineral in the float on the surface or in creek-bed. The appearance of material derived from erosion is indicative of the character of

the rock from regions higher up. Therefore the bed of the stream, or the hill slope, is minutely examined for fragments of ore, or blossom, and followed as long as mineral is found. If the float or shode boulders are pebbly or rounded, or in vegetable soil, they have come from afar and the lode is not at hand. If the shode is large and angular, it has not come very far, and the discovery of a point beyond which no float or blossom is detected is presumptive evidence of approach to the vein. The lode will be found above the point of discovery, and the prospector will go in the direction of the drainage and thoroughly search the ground.

In high altitudes the oxidation of the minerals in, and the electric manifestations of, the vein outcrops have assisted the prospector by the light playing over them. This is of continued occurrence in Colorado above timber line, and particularly in regions of arsenical veins.

When found, the vein should be examined, and its value confirmed at several points; most grievous disappointments have ensued from testing of the lode at one point only. If the country is stratified, care is taken to ascertain all the data of thickness, etc. Frequently the ore oxidizes and rots away, to be crushed by the overlying strata, showing only in a small streak; or the outcrop may fold back, "tail out," and give false impressions of great thickness.

Geological Aids to Prospecting.—Maps are serviceable as showing the important features, and a systematic plotting of all data, geological and otherwise, is necessary. Dr. H. M. Chance, in the Second Geological Survey of Pennsylvania, has an admirable discussion on the construction of geological cross-sections, to which the reader is referred. Prospecting for oil or gas is speculative. Drill-holes must be carried down to the oil-bearing sands, and there is no surface guide to determine the site for a well to be bored within the known limits of a given field.

Explorations.—If the surface fails to reveal the mineral body sought, and there remains reasonable expectation of finding it, a tunnel, a shaft, or boring may be resorted to. The two former are more expensive but safer guides than that offered

by boring. Shafting is slower and more costly than tunnelling, but more quickly reaches a flat seam at a point suitable for development. The steep pitching vein is perhaps best reached by a tunnel, if the depth of vein so gained is great enough to compensate for the length of tunnel. The choice between them depends upon local conditions. Both are advisable for shallow explorations, while drilling may be employed for deep work. The latter is very commonly employed on account of its cheapness. But even when it has determined the data previously doubtful, the shaft or tunnel must subsequently be driven. So drilling has its limitation of use. It is rarely employed as a seeker for mineral, but merely to give confirmation to, and assist in a rational estimate of, the value of the undertaking. Many properties owe their rehabilitation to the results of the diamond-drill exploitation, and none should be abandoned until after a careful surface examination has been made and followed by numerous bore-holes.

Either the punch or the diamond-drill method may be used for the boring. The former is cheaper, but the pulverulent material brought up by the sludger is unsatisfactory; it may indicate the constituents of the rocks pierced at different depths, but can give little of its physical character or dip. The diamond-drill core yields a little more information, but even its indications are hardly trustworthy. It affords an opportunity to identify the rock, but some of the soft strata are worn away or the core may be turned in its tube, so its revelations are not much better than those of the sand of the punch-drill. At best, the results obtained from either cutter are not conclusive, for it may have just missed the mineral, or have struck a solitary, small, soft chunk of ore, which would supply cuttings to discolor the sands for a long distance and give amazing report. Very important deductions cannot be based solely upon the indications of the borings. Only after numerous holes have been bored and a thorough surface examination has been made can a conclusion be reached.

Good, hard common-sense, observation and pluck win, and they alone. There is no mystery about the finding of mineral

deposits. Nature is bountifully supplied with precious metals and valuable minerals, but her secrets are hid. Only the cumulative information of geological experience gives any clue as to the habitat. Neither witchery nor magic charm can disclose the whereabouts of an ore body or deposit.

Divining-rods.—The wizard with the hazel wand, or the spirit medium who is controlled by some disembodied Comanche chief, is an impostor. He affects a versatility and occult power that transcends combined scientific knowledge, but to a paltry amount of "filthy lucre" he is not averse, when he plays upon the credulity of natures which are duped into making extensive explorations upon his purported previsions. This would be ludicrous, were it not also painful, to see the number of misguided men who have squandered hopes and possessions in their search for a short cut to wealth.

The Proof of a Vein.—The discovery of mineral at the surface must be followed up by the disclosure of the vein. In order to secure possessory title to it, the United States laws require the proof of the existence of a lode or vein. Test pits or shafts are sunk to reveal the mineral in place, having a definite direction of outcrop. A single shaft is not regarded as sufficient evidence of a vein. The mineral must be in place. Even if disintegrated near the surface, it is still a vein if a crevice prevails carrying mineral matter between rock of a nature and origin different from it. The U. S. statutes divide mineral ground into veins and placers only, hence the presumption would be that any well-defined metalliferous crevice capable of ready identification by the miner is a vein, whether fissure or not,—only it cannot be a placer.

The Extra-lateral Rights.—The difference in the grants under the two cases, besides a difference in acreage, is that the mining of ore within placer ground is confined to the vertical planes through the boundaries (sec. 2329, U. S. Rev. Statutes), while vein deposits may be pursued along their dip, "throughout the entire depth," even if they "so far depart from the perpendicular" "as to extend outside of the vertical side lines of

the claim"; and the extent of the miner's right is determined only by the vertical planes through the *end lines*, which should therefore be properly drawn. Extra-lateral rights are, therefore, accorded to vein locations.

The Dimensions of a Claim.—The prospector announces his discovery by recording the fact, together with the data of discovery and the direction of the outcrop of the vein, upon a stake placed at the property and upon the record books of the county. This done, he may proceed to work.

Locations 1500 feet in length are permitted upon the *public domain* to the discoverer of the lode. But for access thereto, and for convenience of working, the U. S. grants, as incident to the principal feature, surface ground which, measured from the middle of the vein, shall not exceed 300 feet on either side. Some States have reduced this to 150 feet on each side, while in some Colorado counties only 25 feet was, and 75 is, the outside limit. The claim must be essentially a parallelogram. It may be 1500 feet, or less, in length, located substantially along the middle of the apex, across which are drawn two parallel end lines and side boundaries, within the limit prescribed, parallel in pairs following the contortions of the outcrop. However else the Act may be vague, it certainly is not upon the fact of the parallelism of the exterior boundaries. Excessive locations are valid as to the legal limit and void as to the excess.

Maintaining Possessory Right.—It is incumbent upon the locator to define the boundaries of his claim, by placing stakes at all corners and intersections, to notify others that the ground is entered upon and being exploited. These, with the filing of a location certificate in the county, maintain possessory right from the moment of posting a location notice of discovery upon the lode. Within a reasonable time thereafter, sixty days usually, the locator is required to sink a "discovery" shaft at least 10 feet into the vein. This satisfies the regulations regarding discovery, and maintains a mining right against all comers until the expiration of the calendar year.

From that time on, an "assessment" of $100 must be ex-

pended annually as evidence of mining intent. A failure to expend such sum constitutes a forfeiture, by which the claim reverts to the public domain, and is subject to relocation.

A prospector is not confined to a single entry upon a discovered lode. He may appropriate as many claims as he chooses, contiguous or otherwise, with that of the first discovery. Upon each 1500 feet, or less, of length he must show the intent to mine, by a discovery shaft and the assessment work.

For the development of the mine, the annual assessment work may be done upon the surface or upon the vein, and all efforts outside of the limits of the location with a bona-fide intent to work the claim are justly considered as if upon the claim—as, for instance, development by tunnel instead of shaft.

This concession is further extended by the U. S. Supreme Court; for where one person owns several contiguous claims capable of being advantageously worked together, one general system of development may be adopted, after the discovery shafts are driven. This encourages more economic work and subserves the best interests of all concerned.

The Application for Patent. — When development work to the value of $500, exclusive of buildings, is completed within five years, by an annual assessment of $100, he may proceed to obtain the "patent" or legal title from the government. For this purpose an approved survey of the property claimed by him is necessary. An announcement of his intention to apply for patent is placed on the property, in the local land office and in the newspapers, for sixty days. If no opposition is filed against this request to the government, a patent will be issued to him when he shall have proven the previous facts, and that he is a citizen of the United States and has paid $5 per acre for the land enclosing the vein.

According to the laws, the applicant may obtain as many claims as he desires, provided he satisfies the requirements for the annual assessment prior to making application to the government.

Chaotic Condition of the Law.—The government has been exceedingly liberal to its citizens in throwing open the mineral

lands for exploration and occupation. The original intent was to legalize the possessory system, which had grown up in the absence of Federal legislation, by which the possessor claimed a vein for mining purposes and was allowed the use for right of way to the surface overlying his mine. This entailed the granting of the mineral vein free but the enclosing non-mineral land by sale.

An exclusive right of enjoyment "of all veins" cropping inside of the boundaries is given with the claim. If they are discovered and entered upon in adjacent territory, the subsequent locator, according to the laws of the State of Colorado, may have right of way through the cross-vein to his ground on the other side of the prior claim, but none of the mineral. In every case it is intended that priority shall govern. Sec. 2326 grants to the senior locator the mineral at the intersection, and to the junior the right of way through it.

By the interpretation of the U. S. Statutes, easement and title were clearly intended to be conveyed for all forms of metalliferous deposits, in the use of the terms "veins, lodes, or rock in place." The Act recognizes any mineralized rock in place, enclosed in the general mass of the mountain, as a vein.

The Statutes favored the miner and assumed to cover all lodes whose indications were sufficiently marked for the miner to continue explorations thereon. A crevice, crevice matter, a fair wall, and mineral are the essential conditions.

It has been seen that a lode claim, whether patented or not, carries with it all that is beneath the surface-ground claimed, with a servitude upon the adjoining territory obtaining the right of following the dip of the vein, and subject to a like easement granted to the locator on adjacent ground to pursue his vein wherever it may go. This obtains until some one can show a better right. The common law as to realty is modified when applied to mining property.

It has, however, happened that rulings were so made and construed that a party may locate vacant ground and maintain ownership to the mineral covered by it, unless *it is shown that*

the mineral body belongs to a lode cropping elsewhere within legally claimed ground. The proprietor who calmly continued work upon his discovery found himself breaking into the subterraneous workings of others who had stolen a march on him. To secure his right he had to bring action to eject. To vindicate his title he had to prove the lode to be in place and continuous from the point of his discovery *to, into, and through* the ground of the trespasser. Failing to do which, his claim was defeated and with it all incidents thereto attached.

Naturally the train of reasoning led farther and farther away from the original intent of the law to reward the discoverer of an apex, until the accepted idea is that, although the "defendant's location may appear to you to be along the line of the top, apex, or outcrop of the vein, it cannot prevail against a senior location on the dip of the lode."

To what absurdities the law has led us, by reason of the vagarious interpretations, the reader may learn by referring to Dr. R. W. R. Raymond's articles in the Transactions of the American Institute of Mining Engineers.

The remedy is to repeal the present enactment, or else to so prescribe and define the subjects of the United States grant that purchaser shall have a warranty title to the entry. The side-line law of Leadville is far preferable to the present uncertain grant of extra-lateral right. The risks of mining are sufficient without adding this unnecessary one. It is singular that, whatever amendments have been proposed in recent years, they persist in retaining the essential feature of the extra-lateral right depending upon the apex. So long as this basis is retained they will do no good. There is no reason for longer maintaining this abnormal and indefinable privilege in the grant of mineral land made by the U. S. Government.

Justice W. E. Church, in a concluding and conclusive sentence of a decision, said: "The present laws are a hotbed of litigation and a fruitful source of error." Judge Bradley declared them "imperfect," and those who have had any experience with them will agree.

CHAPTER II.

PREPARATORY WORK.

Mine Development.—Assuming that the question "Can it pay?" has been answered affirmatively, the next feature to be considered comprises the extent and character of the surface buildings, the nature of the preparatory work, the exploitation plan, and the machinery to be installed for hoisting, pumping, ventilating, and for the treatment of mineral.

The Buildings at the Mine Mouth.—These comprise the structure for the power plant and such additional buildings as are needed for the preparation of the mine product. If the latter be pure, or rich enough to be marketable, or if the coal is to be coked, a simple storage building is sufficient. If it is to be sorted, crushed, or screened and shipped, a tipple or a breaker will be required. If the mineral requires treatment and the local water-supply be deficient, or too acid, the washery, stamp-mill or concentrator will be located elsewhere and the mineral delivered to it by tram.

Collieries provide standing room for a day's run-of-mine cars, a storage track for empties and a side track for handling the railway cars, wagons, or coke-larries. If the coal is screened, three or four tracks, side by side, are connected with the main track at both ends.

All buildings should be as nearly fire-proof as the risk demands and the first cost will permit. A large colliery with, say, 1000 acres of ground to work, or a well-developed metal-mine, will thus erect very substantial buildings. These should be as near to the shaft as safety to the mine will allow and yet be remote

from the mouth, as far as is consistent with economy. The hoisting engineer should have an unobstructed view of the dumping platform, and his engine should be far enough away to give the rope a good fleet angle. The hoist-frame is open and not housed.

The Selection of a Site.—The location for the offices, the buildings, and the point of shipment for the mineral must satisfy the surface conditions, must be convenient to the shipping-tracks, and at the same time one which does not require much grading for the buildings. The location of the railroad and its switches fixes the location of the ore-sorting or coal-screening building. As the surface plant may become quite extensive, its location must be considered with a view to the economy of handling in such a manner that the mineral shall descend naturally in its progress through the works. The location of the screens fixes the site for the mouth of the mine. The latter must therefore be in the best position for both the track and the mine.

The opening into the ore body must satisfy the underground conditions of haulage and drainage, and it must be centrally located to the lowest point of the vein. It would be an ideal condition if the shaft which would satisfy the shipping requirements could also be placed in the centre of the basin with the haulage and drainage toward it from all sides. As this is not likely, some compromise must be made between the various demands of surface construction and mine operation.

Choice of Character of Entry.—The method of attacking the vein, or bed, varies with the inclination of the mineral body and its depth. Undoubtedly, when the mineral occurs at considerable depth, the sole method of attack is by a vertical shaft. If the depth be not very great, a slope from some convenient point may be preferable. When the vein, or the coal-seam, crops out at the surface, within the property lines, an entry to the lower depths is obtained by a slope following the inclination of the vein. If the outcrop so exposed is quite long and extends along the side of the hill, several slopes may be driven to advantage. When the outcrop occurs on the hillside, the vein occupying a position

somewhat vertical, undoubtedly the best method of entering the vein consists in driving one or more horizontal openings, or adits, into the vein at various heights according to the system to be employed in the subsequent explorations.

Owing to the difficulties and the expense of sinking shafts and of hoisting the material through them later, conditions sometimes arise when it may be preferable to drive a tunnel from some point as far below the outcrop as possible or desirable and toward the vein with the shortest entry. Such a tunnel is called a cross-cut or a cross-country tunnel, and in some regions is known as a rock tunnel, the significance of the titles being easily understood.

Certain conditions in coal-mines warrant the driving of rock slopes as a compromise between the horizontal tunnel and the vertical shaft. They are employed where only a shallow shaft would be required, but where a high speed of hoist is regarded as desirable.

Naturally the endeavor is to secure the most direct line to the point of exploitation, within the shortest time and at a minimum cost, while at the same time constructing a permanent opening which would serve for all the work during the life of the mine. The opening of the mine is near to the mill-building where the mineral is screened, weighed, washed, or treated, preparatory to being marketed. The character of the opening can be decided only after knowing the exact nature of the underground conditions of the vein. Its extent, thickness, depth and general inclination must be known. Some clue to this can be obtained by prospecting the entire field with the use of bore-holes and open cuts, and geological maps are exceedingly serviceable, but in metalliferous districts it is hardly possible to foresee the character and lay of the ore body. Without this full knowledge, it is likely that the opening made for the mine during exploration would later be abandoned as being in an undesirable position or too small to meet the final demands upon it. In coal-mines the eccentricities of the vein are not so pronounced as to give the same difficulty to the engineer.

Coal seams have less irregularities than do the metalliferous veins. Their contents are quite uniform in character, and the thickness of the seam varies to a less degree than does a vein. Both are found dislocated by faults or shattered by movements of the strata. The metalliferous vein, however, not only changes in thickness and inclination but its mineral contents will vary in percentage and even in chemical composition. Frequently the mineral at a lower depth bears no resemblance to that at the surface.

The character of the mine opening depends upon whether the ore body or bed of coal is revealed at the surface or not, and whether or not the outcrop is included within the boundaries of the property. It may occur on the hillside; its strike of the outcrop may follow the hill, nearly horizontal; or it may lie in the direction of the slope somewhat approaching the vertical. If there is no outcrop the depth of the ore body is known.

The Mine Opening—The entry into the mine may begin on the mineral at its outcrop and, continuing, follow it wherever it may go; or the opening may begin at some point on the surface away from the vein to intersect it at a selected point.

The openings are horizontal, vertical, or inclined, the choice between them being determined by the conditions named above. The horizontal openings comprise drifts, adits, and tunnels. The vertical openings are designated as shafts. The inclined openings are termed slopes, or inclines. The open cuts are either quarries or strippings.

Metal-mines are operated usually from one entry or opening. Only under certain conditions presenting dangers of caving and other reasons requiring the hasty removal of the mineral is it desirable for metalliferous mines to have many openings. Coal-mines are required by law to maintain two parallel openings at a certain stated minimum distance apart. These are driven simultaneously or upon the coal-seams.

The Preparatory Work.—The operator of a coal-, clay-, or lead-mine is confined, in his choice of site for the shaft, or tunnel, within the property lines. Metal miners, in regions

other than the national domain, are also confined by the vertical planes through their boundary-lines.

On the national domain the miner is not restricted to his own property in choosing the site or the character ofen try for his mine. His property is bounded by vertical planes through the end lines, but not through the sides; and he may therefore operate his vein on an inclination to any depth desirable. He is not confined to his surface boundary lines when selecting a site for the point of attack upon the vein. He may sink a shaft from outside the boundary-lines, or drive a tunnel from the foot of a hill to his vein, provided only that no other existing mining rights are injured thereby or are interfered with. A greater latitude of choice is therefore afforded the precious metal miner than the coal-operator. Under such conditions, the tendency is naturally to drive the horizontal cross-cut tunnel to the vein as being the most economical method of developing the property. The great expense and the changes in the character of the vein militate against using a cross-cut tunnel, notwithstanding the facilities it offers for shipment. It is a question whether they would outweigh the importance of "staying by the mineral."

Adits.—Veins cropping out on the side of a hill are often exploited by driving adits, which are merely horizontal openings following the vein. The cost of driving is less than that of a slope or a shaft. The cost of equipping is slight, and it serves all purposes of permanent hauling, drainage, and ventilation, as well as during explorations. These adits are driven at distances apart vertically, depending upon the thickness of the vein and the facilities for shipment. They block the vein into sections, each one of which above the adit can be mined at a very low cost. Their dimensions, equipment, timbering, and gradient are the same as those of underground galleries.

Cross-cut Tunnels.—These have all the advantages of the adit in furnishing a convenient haulage and drainage-way at a point far below the average of the workings. They furnish a cheap, secure, and permanent entry if the mine has been developed.

It is a question, however, if a long tunnel should be driven as a means of exploration before the mine has been carried to the corresponding level that the value and character of the mineral may be known.

Vertical Shafts.—These constitute the most secure, but the most expensive, method of entry into the mine. But the shaft presents many of the objections which also obtain against the tunnel. The shaft is the only means of economically reaching a coal-bed which is not shallow and whose outcrop does not occur within the lines of the property. For reaching veins this is an uncertain method of attack, particularly if the vein changes materially with depth in its inclination and character of mineral. The great expense of sinking a shaft requires it to be carefully located in a position which is certain to be that of the permanent outlet of the mine.

If a choice of position be afforded the engineer, the foot-wall side of the lode is usually selected as being safer than the hanging-wall side. But each successive lower cross-cut is longer than the one above it, and if the country rock be hard and the vein very steep, the cost of these cross-cut drifts would soon become prohibitory. For this reason a majority of shafts are sunk from the hanging-wall side to intersect the vein at a moderate depth.

The size of the shaft is as great as the capital will allow, though the tendency is to restrict its area. The dimensions of the hoistway depend upon those of the mine cars, which in turn depend upon the character of the underground haulage systems and the dimensions given to the haulage-ways. The width of the shaft compartment is equal to the length of the car plus 6 inches at each end. This direction is the long side of the compartment and is laid parallel to the mine tracks and also to the straight track to the tipple. The shaft may have only two compartments, both being used for hoisting; three compartments, in which the third is for timbering, piping, etc.; or even more compartments, the last being for ventilation. The other dimension of the shaft will be determined when the decision has been

reached as to the number of cars to be run upon each cage at a time and the number of compartments required. The capacity of the shaft is determined by the conditions discussed in Chapter V; the dimensions of the shaft, the method of sinking and timbering, in Chapter I of Part II.

Inclines.—Inclines and slopes are in great favor for many reasons. They usually follow the vein and explore it, thereby giving a return in the mineral recovered for the cost of driving The maintenance of slopes is, however, higher than the cost in shafts. Slopes are provided in coal regions when the inclination is over 10° and the depth to be reached not over 500 feet.

When the vein is constant in pitch and has no irregularities, everything favors its driving. If, however, the vein has occasional enlargements or a varying pitch, its pursuit becomes awkward. The question then arises as to whether it is desirable to follow rigidly the irregularities, to reduce them to uniform pitch or to abandon the entry altogether for a shaft or tunnel. Only the local conditions will settle that.

Blocking Out the Ore Body.—The preparatory works are far from complete when the mineral body has been reached. A flat coal-bed or a slightly inclined vein is treated, for the purpose of mining, as a tabular mass of mineral which is to be divided into blocks by haulage-ways. The latter are called galleries, or gangways, in coal-mines, and levels or drifts in metal-mines. The product from each block is delivered down grade to its gangway, whence it is hauled to the slope-shaft or tunnel to daylight. These gangways are as numerous and driven as close together as the requirements for exploitation will demand or the means of the operator will permit. The more numerous they are the more quickly can each lift above them be exhausted. On the other hand, as they cost more to excavate than the corresponding volume of material in the rooms, this character of work must be confined to its lowest limit. Between these two conflicting requirements, each mine presents its own local conditions.

Dead-work.—The gangways, or drifts, partake of the nature of explorations, and, being more expensive, and in many cases

only tentative, are termed dead-work. By dead-work is understood any bore-hole into the vein, auxiliary shaft, drift, or gangway and the narrow work of connections between the rooms. Though primarily unproductive, dead-work bears a vital relation to the economy of the mine. The location of the gangway is a matter of great importance. If the mineral is of varying grade of value and soft, it is laid along the foot-wall. It is in the country rock, near the vein, if the former is softer than the vein matter. This enables the seepage and the drainage to be taken care of and reduces the risk of injury from subsidence.

Lifts in Metal-mines.—Metalliferous veins are blocked by driving levels, or drifts, at distances of from 60 to 100 feet apart, measured along the slope. These are started on either side from the slope or from each cross-cut intersection with the vein and carried along the vein. The casual connections made between the adjoining levels occur only when the working of a lower lift has carried that opening to the level above. The ventilation requirements in metalliferous mines are not sufficiently urgent to require the continuous circuit of air through the various openings in the mine. The height of the lift between the levels and their ratio to the thickness of the deposit depend more upon the percentage of the mineral of the vein contents than upon the method of mining. In soft ore the levels are close together and the lifts not greater than 60 feet. In hard rock, in thin deposits and in ore of low grade, the distance between the levels is greater. The maximum, however, is 100 feet.

The Lifts in Coal-mines.—The coal-bed is divided into lifts of 300 feet to 600 feet along the slope, the greater length for flat beds. Double entries are started upon an inclination from shafts, or slopes, with such grade as depends upon the system of haulage. These gangways are driven side by side in pairs with an untouched rib of coal of at least 20 feet between them (Fig. 7). Their direction may also be indicated, if not determined, by the cleavage planes of the coal. One entry in each pair of gangways serves as the intake airway to its lift of coal, and the other for the outgoing air-current.

In thick and steep coal-seams the haulage-way is usually built near the floor to facilitate the landing of the cars and the parallel returning airway similar in area, above it or nearer the roof. From these gangways the rooms are driven upward on the slope, at distances apart of about 40 feet, to the gangway above.

The Dimensions of the Horizontal Opening.—The dimensions are governed by the service which the tunnel, gangway, or adit is to give. They are not unnecessarily large, for, in addition to the great cost of excavating, there is added the additional cost of supporting the increased opening and the expense of maintaining it. If it is to serve as a ventilating way only, its area is as great as the requirements of the ventilation systems demand, the height and width depending on the special conditions. If it is to be a haulage-way, a single track is provided, unless the number of cars passing through it for a given output demand a double track. The latter would probably be used only for an endless-rope system of haulage. The height of the gangway is made just sufficient for the passage of men through it and only as much more as is required to furnish the area necessary for the ventilation current, provision having been made for the probable interference with the current by moving cars. The width must be sufficient to provide ample clearance space beyond the cars for the safety of the men walking along the way.

Reserves.—The number of blocks opened at a time is only enough to maintain a regular output. Whatever the method may be, it aims to concentrate the men as much as possible, while giving them working places of ample size to break down the mineral over as large an area of free face as possible, to reduce the length and cost of gangways to a minimum, leaving them open only as long as needed, and robbing the supports as promptly as the rooms are abandoned.

The distance to which the gangways are carried beyond the shaft or slope depends upon the relation between the cost of their maintenance and of the haul to that of sinking an entirely new entry some distance. A mile is as far as ordinary condi-

tions of haulage allow; beyond that, a new opening is necessary. But to maintain a constant output for the demand of the market, the levels, or gangways, are kept in advance of the immediate places of work. Their revelations furnish a guide for future action. This is particularly desirable in precious-metal mines, the value of whose mineral contents is so variable. These advance blocks, together with those of exceptionally high grade or of medium quality, previously developed, constitute the mine reserves. When the vein rock is soft and requires considerable timbering for its maintenance, this character of development work is not carried far beyond the work of mining.

Pillar Supports for Mines.—All permanent ways in horizontal deposits are protected from the crush of the roof to secure a safe outlet. Shafts are surrounded by pillars of untouched vein matter or coal, whose dimensions depend upon the depth to the bottom of the shaft; haulage-ways in coal have massive pillars on each side undisturbed except for the narrow, short openings connecting the rooms with the haulage-ways; and the rooms are protected by pillars of undisturbed coal on either side. These reserves afford abundant support and are undisturbed until the working places they guard are to be abandoned. In veins or nearly vertical beds where the wall pressure on either side does not tend to close the rooms or the haulage-ways, sufficient security is attained by horizontal timbers between the walls above the haulage-way and by untouched blocks of mineral alternating with the rooms (stopes).

The Influence of Cleavage Planes in Determining the Direction of Roads.—All strata are more or less uniformly creviced, with horizontal planes of growth and vertical joints, caused by shrinkage. Coal masses are similarly divided by one or more sets of planes producing rhombohedral coal. These cleavage planes in coal, called "cleats," facilitate the breaking of the mineral. The direction of the rooms in coal-mines is fixed by the direction of the cleat, the important gangways being carried with the cleat. In hard-coal regions, where explosives are extensively used, the cleat does not materially assist the breaking

of the mineral, and as a consequence the rooms are not dependent for their direction upon the cleavage planes. In steep bituminous seams cleat has less importance than grade for haulage-ways.

Faults.—A fault is a break in the continuity of a vein or stratum. During some convulsion of nature the strata were broken and in the readjustment of position a vertical displacement occurred. The plane of fracture may be empty or it has been filled with extraneous matter. When encountered during the operation of mining the fault plane or cross-course is pierced and its strike and pitch noted. On the distant side, the character of the rock is examined. In a stratified country it should be easy to identify the stratum and its geological position relative to that of the ore-bearing stratum. Thus the engineer may be guided up or down according to the direction of the throw. But operations in massive rock assume a more serious aspect. On the distant side of the dike or cross-course no geological aid is available. Hence, in attempting to follow the prolongation of the vein beyond the fracture, the engineer must rely upon his knowledge of the local conditions or upon the average of those prevailing in similar ore-bearing fields.

Schmidt's Rule.—It is a matter of record that 80 per cent of intersected veins were heaved, apparently, to the right or left. Those to the right are twice as many as those to the left. Henwood also discovered that the heaving to the side of the greater angle is five times as common as to the smaller angle. In every district may be found a rule for finding the other end of the vein, but it is purely of local application and unreliable. To formulate a general rule out of these numerous and apparently eccentric displacements would seem well-nigh impossible; but Herr Schmidt, in 1810, offered a solution to the problem, which, though not infallible, is the best extant and has done valuable service. "When the cross-course dips away, after going through it, the drift is run along its far wall in a direction opposite to that in which the vein pitches. If it dips toward the mouth, the drift is carried along the far wall, to the right or to the left, as the veins dip to right or left." The amount of the displace-

ment, i.e., the distance to be drifted for the continuation of the vein, cannot be premised. It varies between very wide limits, and is thousands of feet in many localities Henwood averages the throw of veins at 16 feet.

Open Cuts.—Open cuts are used to explore the outcrops of veins and shallow mineral bodies They contemplate limited explorations and a smaller output and life as compared with that of a mine. They are also employed for stripping the surface coverings of flat veins which are close to the surface. Quarries are extensive open cuts for the extraction of iron ore, slate, peat, building-stone, and such large bodies of soft minerals as would be difficult to support by timbering in the usual methods of underground mining. In quarries, large masses are disengaged at a time and the operations are conducted upon a large face at a comparatively small expense. The entire deposit can be recovered to a moderate depth with a high degree of profit. The point of attack for a quarry is the lowest consistent with the requirements for transportation. It is operated in benches and cut by mechanical picks or channeling-machines (Fig. 16). Hoisting from the pit is accomplished by derrick and bucket with a form of suspended cable-way. The pulsometer or a centrifugal pump is used for drainage. The depth to which a quarry can go is limited to, perhaps, 200 feet, beyond which some more rational method must be introduced. It involves some form of underground work which ultimately becomes far more expensive than the mining which could have been adopted prior to the operations of quarrying. Open cuts are always to be deprecated where a subsequent underground work is to be pursued.

The objection to quarrying is the damage which is done by the influx and accumulation of surface-waters. These give trouble to the miners and must be removed as promptly as possible to admit of sinking of the quarry as readily as the output demands. Again, the difficulty of propping the sides of the cave is no small matter, particularly if the deposit occurs between pronounced walls with an overhanging side. The Tilly Foster

mine was deepened only by the blasting away of 200,000 tons of the overhanging wall. This character of quarry work is dangerous and not economical, but is universally employed for iron ores. In the case of the diamond ground of the Kimberley mines, the quarry was replaced by a method very closely approximating the long wall for coal and caving of iron ore. The surface-waters accumulating in the open quarry, which found their way into the mine, were constantly threatening danger, and the operators finally dug tunnels around the quarry to catch and divert the surface-waters.

The Steam-shovel.—In soft minerals the steam-shovel with its dredger derrick is employed for excavation purposes. Its introduction in the iron-mines of Wisconsin and Minnesota has enabled the marketing of iron ore of a very low grade at low cost. The cost of quarrying, crushing, and administration of the iron-ore deposits in the northwestern portion of the United States and in British Columbia rarely exceeds 90 per cent per ton, and in many cases is very much lower. Some of the copper deposits of British Columbia are also of such phenomenal size that their development by the steam-shovel has brought about the introduction of machinery for hydraulic-, electrical-, and steam-power purposes in one case up to 6000 horse-power. Air-machine drills of $3\frac{1}{4}$-inch size are used as auxiliaries where the rock is sufficiently hard to demand their employment upon the immense boulders which are interspersed with the softer rocks. The quantity of earth which can be removed by a steam-shovel varies, of course, with the supply of wagons which it has and the length of their haul. But a 45-ton Bucyrus shovel having a bucket carrying 1.75 cu. yds., excavated in a day about 1200 bank yards from a cut whose depth was 8 feet and whose average width was 22 feet. In this case automatic dump-wagons were employed, and with a haul averaging 1500 feet each way. Three wagons were filled per minute.

Hydraulic Mining.—This is a species of open work in which water is the agent for cutting and transporting the gold-bearing earth. From nozzles at the lower end of pipe lines, leading from

reservoirs located at convenient points of great elevations, the water is discharged at a high velocity against the bank of a stream or an old river-bed to undermine it and to wash the material into sluices conveniently placed to receive the flow. The sluices are built on an inclination, with cleats on the floor and of a width and depth sufficient to carry the volume of water and its suspended material. The later installations of hydraulic plants use centrifugal pumps (Fig. 122). In two or three steps the water is raised from the creek and delivered to the bank at high pressure and velocity.

Hydraulic mining is inexpensive because of the small amount of labor involved. Four men can operate the entire plant, hence the process can be conducted at low cost and with a high degree of efficiency. Earth carrying 6 cents in gold per cubic yard is regarded as "pay-dirt." The total cost of operation is \$0.082 per miner's inch. About 2½ to 3 cu. yds. are moved per miner's inch of water used.

A long sluice is unnecessary. All the coarse gold, and nearly 70 per cent of the total fine gold, recovered is caught in the first section of boxes, about 12 per cent being obtained in the second and an insignificant amount in the fifth section. The grade of the boxes is 5 inches per foot.

The Duty of Giants and Lift-tips.—The following table shows the distances which the standard nozzles can throw a stream of water together with the quantity of water which they can handle under the stated pressures.

Salt-mines.—The getting of salt is generally by a special process. It is always found in old river-beds and frequently quite thick. The process of mining follows one of two methods. Pure thick beds are mined systematically by a method of squares or like similar bodies of soft coal, due regard being had to soundness of roof and solidity of the mineral. When the salt body is thin or somewhat impure, a different method is employed. Holes are drilled to bed-rock, water poured down and the rock-salt bleached out. A pump-pipe line is extended to the floor level and the brine raised to the surface.

DUTY OF GIANTS.

Diameter of Nozzle, Inches.		Pressure at Nozzle.							
	Pressure in pounds	30	40	50	60	70	80	90	100
	Head in feet	69.3	92.4	115.5	138.6	161.7	184.8	207.9	231
1	Gallons	134	155	173	189	205	219	232	245
	Horizontal distance	90	109	126	142	156	168	178	186
	Vertical distance	62	79	94	108	121	131	140	148
1⅛	Gallons	170	196	219	240	259	277	294	310
	Horizontal distance	93	113	132	148	163	175	186	193
	Vertical distance	63	81	97	112	125	137	148	157
1¼	Gallons	210	242	271	297	320	342	363	383
	Horizontal distance	96	118	138	156	172	186	198	207
	Vertical distance	63	82	99	115	129	142	154	164
1⅜	Gallons	253	293	327	358	387	413	439	462
	Horizontal distance	100	124	146	166	184	200	213	224
	Vertical distance	64	83	100	118	133	146	158	169
1½	Gallons	313	361	404	442	478	511	541	571
	Horizontal distance	104	130	152	176	195	212	224	240
	Vertical distance	64	83	101	119	135	149	163	175
1¾	Gallons	425	490	548	600	649	694	735	775
	Horizontal distance	110	140	161	188	210	230	243	259
	Vertical distance	65	85	104	121	138	154	169	183
2	Gallons	556	642	717	785	849	907	962	1014
	Horizontal distance	116	147	166	194	218	240	255	274
	Vertical distance	65	86	105	124	141	158	174	189

Peat is recovered by dredging channels usually cut in the heavy bogs deep enough to serve for drainage and for navigation. The peat is then cut in blocks. When it occurs above water the fuel is quarried in steps as are building materials.

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CHAPTER III.

METHODS OF MINING.

Exploitation.—The plan pursued for winning minerals and making them available for use is termed *exploitation*, while the general term *mining* is applied to the labor of excavation.

For the purpose of exploitation mineral deposits may be classified according to their shape and inclination into those which are of irregular shape and those which are tabular in form. The former are mined by *open cut*, quarry, stripping, or the hydraulic process, with or without the aid of a dredger or steam-shovel, as they are usually superficial in position and their contents friable or soft. The latter are composed of comparatively hard material and are extended in a horizontal or vertical direction with some degree of uniformity. Their contents are recovered by some system of *Underground Mining*.

The character and depth of the superficial covering may determine the choice of method for some deposits which occur in tabular masses. The relative convenience and cost of removing the cover, and the increased expense of supporting it above the subterranean works, constitute the decisive element. Open-cut mining may be profitable when the quantity of superficial material to be stripped from the surface does not exceed one half of a foot depth for each foot of thickness of material in the deposit, and where the quality of the cover is such as can be easily removed by the steam-shovel.

Open-cut Mining.—The method of extracting the iron ore in the Mesaba range in Minnesota as adopted by the Iron Mining

Company consists in stripping the surface material by a steam-shovel of the dipper type. This machine has a radius of action of 15 feet and excavates and delivers the dirt to cars of the standard railroad gauge. A cut is first made for a certain distance and width in which the track is laid to admit the machine. The steam-shovel excavates the dirt on one side of the cut, delivering it on the other side, and stripping the surface for the depth of cut to the boundaries of the property. The shovel is returned to its starting-point, and from the side cut where the dirt has been stripped a new track is laid and the work proceeds, while cars are loaded on the first track. The stripping advances over the entire area to the level of the track. Several steam-shovels strip in a similar manner until the ore body has been exposed. The usual depth of a cut is 20 feet; beyond that there is danger of the bank caving and causing injury to men and machinery. Occasionally deep holes are drilled and chambered into the bank ahead of the machine, into which several kegs of powder are loaded and blasted for the purpose of loosening the ore for the work of the shovel.

The attack upon the iron ore proceeds in the same manner as in the surface material—in benches of a width of 30 feet, containing one track for cars and a short section of track for the machine. The product per shift of ten hours is 1800 tons, though there are records of a daily product of 3544 tons.

The cost of stripping and wasting the mineral of a coal-bed, under the most favorable conditions, is approximately 20 cents per cubic yard, whether machines are employed or hand labor alone is utilized for extraction. The sole advantage in the introduction of machinery is the rapidity with which work can be prosecuted and the increased area of stripping which is possible. These are the determining features in situations where the cover is thick and the working season short. Inasmuch as the railroad approach to the several benches on which operations are conducted must also be lowered with the progress of the work, it soons happen that its grade becomes excessive and the traffic is interfered with unless a corresponding increase in the length

of the approach be made. This necessitates a deep open cut for extensive work, or a tunnel through which the railroad cars may be drawn to the works.

The Milling System.—When a limit has been reached for economical depth in the quarry an underground process may be introduced in combination with the open cut. This is termed *milling*, and has been practised to advantage in the Lake Superior iron region. A mill is a hole made in the rock or ore connecting an upper with a lower level of the deposit, through which material is sent down to the level below. In the system mentioned above the tunnel is driven large enough to admit railroad cars into the deposit, which, at distances of two car-lengths apart, is connected with the surface by mill-holes sunk from the surface or from the level above previously worked by steam-shovels. At the bottom of the mills are chutes for loading into cars the ore blasted down at the surface and delivered through the mills, to be thence hauled out of the tunnel.

This system of mining is very successfully loading a train of twenty-five 20-ton cars per hour.

Stripping Anthracite Coal.—A lenticular coal basin at Lattimore, Pennsylvania, is attacked by stripping and underground work. The depth at the centre of the deposit is about 100 feet. After stripping the earth cover from the coal a slope is sunk on the foot-wall of the basin to the centre or lowest point of the deposit. Levels are driven along the strike, and rooms turned right and left, which are worked upward to the surface, the coal being allowed to fall to the level below. Here it is loaded, trammed to the slope, and hoisted out of the mine. The product of the mine is limited only by the facilities for transportation.

Underground Mining.—The deposits having a tabular form are of two types: beds, or seams, occurring nearly horizontal, and veins whose inclinations approach the vertical. The former extend laterally over a considerable territory and under conditions which are quite uniform throughout, permitting a system of mining to be promptly selected. Veins present so many eccentricities which increase with the progress of the work that the

early operations are somewhat tentative until the property has been thoroughly explored.

Deposits of coal, iron ore, and minerals occurring in the stratified rocks are, with rare exceptions, flat. They have a greater regularity of thickness, inclination, and character of contents than do vein deposits in the metalliferous regions which are encased in hard rock, occupy a vertical position, and are subject to changes not only in dimensions and depth, but even in the character of their mineral. The brittleness of the two minerals is markedly different. Coal must be delivered in large sizes for the market, while metalliferous ores are just as valuable when pulverulent as in the lump. The entire contents of the coal-bed can be utilized and are salable, and hence are extracted as completely as possible. On the other hand, in the precious metal deposits not one tenth of the contents between the walls is of sufficient value to warrant hoisting to the surface; the remaining nine tenths, being valueless, must be stowed away. These radical differences in the character of the deposits require distinct systems of extraction for coal and for ore, and hence they will be treated separately.

In one respect only is the mining of coal and the mining of other minerals conducted alike. The operations progress to the rise or upon the slope except on exceedingly rare occasions, when precious metals may be worked downward.

The thickness and pitch of the mineral deposit, the thickness and nature of its cover, the direction and its outcrop, the mechanical and physical character of its rock or mineral having thus been ascertained, the place of opening and the character of the entry into the mine are selected according to the principles discussed in the previous chapter. The directions of the gangways to be used for haulage and of the rooms for economical work are next determined in connection with the system to be adopted. After this the vein or bed is blocked out and is ready for mine operations.

Systems of Mining: Beds.—According as it may be desirable or necessary to support the stratum above the mineral bed or

allow it to fall, the mineral should be removed in two distinct operations, or should be extracted in its entirety at once. The first is the *pillar-and-room system*, in which chambers or rooms are cut out at various points, isolated from one another by untouched portions of the bed, and the mineral from them extracted on the advance from the shaft or entry toward the boundaries. The remainder of the deposit is attacked on the return. This system is applicable where the roof is sufficiently strong to be left with a partial support.

The second, *longwall system*, contemplates the extraction of the entire contents of the bed in one operation. It is employed when the roof is treacherous or weak and cannot be economically secured. No effort is then made to support it. By this plan the entire width of the coal property is attacked simultaneously; or mining may progress circumferentially and outward to the boundaries, removing the entire mineral contents; or all coal may be removed while progressing in the opposite direction from the boundaries to the shaft. These methods are known as the *longwall advancing* and *longwall retreating systems*.

Naturally the simplest method of attacking the mineral would be to complete a series of breasts as wide as the nature of the roof or the walls would permit, and opening them when the main headings have been carried to the extreme boundaries, leaving the mineral intact except for the necessary roads. The breasts are mined out as rapidly as possible on the return homeward toward the shaft. This plan of mining retreating represents the maximum degree of safety for miners, who are then working in solid mineral. It is economical for operators, who may recover the entire deposit. The cost of mine timber is far less, and the product per acre greater, than by mining outward from the shaft. But the plan requires patience and a large capital, particularly if the area to be mined is large or the royalty to be paid is great. It cannot be unconditionally recommended.

Systems of Mining Veins.—The systems employed in recovering mineral from veins are essentially the same as for beds with such modifications as are due to the increase in pitch. The

rock to be supported is the vein matter itself, while the rock adjoining the vein exerts little pressure upon the vein matter. Hence the character and strength of the vein matter determines the choice of the system. In strong mineral and hard coal the pillar-and-room system is adopted. In metalliferous districts the rooms for the first excavations are called stopes, and the system is a *stoping system.* The longwall system as applied to veins consists in the attack of their contents over a broad area and allows the friable materials to cave without restraint. The broken mineral contents are then removed and replaced by additional material caved from above. This continues until the lift has been exhausted. This caving system is much in vogue in iron mines.

Two modifications bearing resemblances to both of the typical systems are also in extensive use, one being known as the *square-set system* and the other as the *filling system.* In these the area of attack is broad, but the roof is not allowed to cave. The excavations are filled as rapidly as it is possible to supply the material for support. In the square-set system a massive cribwork of timbers is built into position. In the filling system foreign materials are lowered into the mine from above, spread and packed into the rooms immediately behind the miners who are engaged in extracting mineral.

Systems of Mining Coal.—The two typical systems employed for extraction of coal are the *pillar-and-room* and the *longwall* previously described. The longwall system is employed for thin seams of coal, or for thick seams divided into thin layers by partings, and having regular thickness, a uniform pitch, weak roof, soft floor, and the bed deep. The inclination of the beds for the longwall system must not exceed 10°.

For the pillar-and-room method the vein may be thick, the material for filling and gob may be absent, the vein shallow, the coal soft, and the seams irregular in thickness and eccentric in pitch.

As not all of the conditions favorable for either system exist in a given mine, the choice must be indicated by the comparative merits of the majority of favorable conditions of greatest

importance from an economical standpoint. No system can entirely satisfy all the conditions; hence a compromise inevitably results in adopting the least objectionable system, with such modifications as have been evolved from local conditions or are based upon the structural difficulties encountered in the given mine. The widths and directions of the rooms or the stalls for attack,* the mode of recovering the mineral from the pillars, and the process of filling and packing the excavations vary in localities and mines and cause a departure from the typical method in some cases to such an extent that the identity is almost lost.

The Longwall Advancing System.—This system consists in driving gangways from the shaft, leaving around the latter an ample pillar for security (Fig. 1). When its boundaries have been reached the gangways are extended, the miners proceeding to the right and left from them, cutting the coal while advancing from the shaft and throwing behind them such waste and débris as are produced during the operations. No effort is made to support the roof beyond that which is necessary for the protection of the men for ten yards back of the face. The gangways serving as roads are carried with the progress of the work by building pack-walls out of the material obtained from the débris. They are advanced radially. Each length of working face is a stall. When two miners are assigned to a stall of about 30 feet each, the gangways or haulage roads serving two stalls 60 feet long are not over 60 feet apart. This necessitates building additional roads to reduce the miners' length of haul.

The numerous stalls extend in one continuous straight line, as in Figs. 1 and 4, or a curved line, as in Fig. 2, without any corners or breaks. The roof either bends, yields, and closes upon the excavation, or it breaks freely and fills the space which is known as the *gob*. The subsidence of the roof is communicated to the overlying strata and will often extend hundreds of feet

* The reader will find in the appendix the definitions of the various technical terms employed in this chapter and elsewhere throughout the book.

Dotted Lines, Progress by Months.

Fig. 1.—The Longwall System in a Flat Seam.

toward the surface. Hence the bed must be deep or the strata firm, if the surface land is not to be injured.

The Scotch System.—A system of mining known as the 45° system, and by some as the Scotch system of longwall, is shown in Fig. 2. The main entries *aa* are at right angles to each other, being turned from the right and left from the upcast shaft *c*. The gangways *bb* are turned at an angle of 45° from the main entries.

FIG. 2.—The Scotch Longwall System.

The shaft pillars *p* are left untouched; the dimensions of each are discussed in Part II, Chapter I. The face of the main entry is advanced with a width of 30 feet, allowing 12 feet between the pack-walls, or 9 feet on each side. Miners turning from right or left of the head of the entry break the coal over its entire face and advance outward, shipping the mineral through the roads. The pack-walls are extended on either side as fast as the face progresses. The roads are so driven as to leave from 120 to 180 feet of face, according to the nature of the room.

Cutting the Coal.—The process of mining the coal is shown in Fig. 3 at *a*, where a groove is cut in the floor or near the floor

by pick or by machine. Two rows of props *bb* are placed parallel to the face and at such distances from it as to give ample support without interfering with the work of the miner. The sprag *c* supports the corner of the underhold coal while the miners are at work. The sprags are removed, allowing the roof to release the coal as soon as the length of the stall has been undermined. No powder is used, the pressure of the roof being employed for the

FIG. 3.—A Cross-section at the Longwall Face.

purpose. The slate and débris are thrown on the gob behind the men; the back line of props is withdrawn and advanced in front; the track is advanced in sections parallel to the face; and the operation is repeated for another 3 to 5 feet according to the depth of the groove. If the roof is weak, little space is left for the track and the gob-roads are nearer together, the stall being shorter.

When the inclination of the coal-seam exceeds 10° the component of the vertical pressure of the overlying strata transmitted by the roof to the coal parallel to the bed of the vein causes a side pressure or "swing" which tends to produce slip in the coal. This is serious, and if the curved face of the stall will not provide for it the system must be abandoned.

Usually the mining is conducted by day, while the construction of the pack-walls and the driving of gob-roads proceeds at night. If it is necessary to operate in two shifts, the stalls are worked alternately from the main entries.

The Demerits of the Longwall System.—The success of the system depends upon the nature of the roof and its behavior when the coal is removed. It should be brittle and weak. A tough

flexible roof is unsuitable and may require blasting down to relieve the pressure upon the coal. The roads which are maintained through the excavations are called the *gob-roads*, and herein lies the serious objection to this system. The fracturing of the roof which is the essential element of success tends also to close the roads, increase the expense of mining, and endanger the lives of the men. The roof and floor are continually being cut into to maintain sufficient height for the travel in the roads. The length and number of the latter increase with the progress of the work, and the cost of their maintenance becomes a serious one. If the boundary is a great distance away, the expense of keeping them open may be so great as to necessitate a discontinuance of the system. The liability of the material in the gob to spontaneous combustion and fire is an additional serious danger. A fire once begun cannot be extinguished without the entire removal of the waste. It is impossible to make the pack-walls air-tight. Ventilation presents difficulties, for the numerous roads must receive separate air-currents, while the working face may easily be ventilated by a single continuous current.

Longwall Retreating.—This is considered a better and safer system, as the roof pressure is supported by the solid coal while the work is proceeding toward the shaft. The waste produced in mining is allowed to accumulate behind the men where the roof closes upon the gob. There is no danger of any section of the mine being closed by collapse, nor is there any fear of spontaneous combustion.

In Fig. 4 is shown a system of opening the mine by longwall retreating. The four pairs of entries *aa* are started from the downcast shaft and driven at right angles to one another to the boundaries between them; entry pillars *cc* are left about 40 feet thick with breakthroughs at intervals of 100 feet apart. When the entries have reached within 300 feet of the boundary gangways *dd* are driven, and at right angles to them the face entries *ee*, at intervals of 200 or 300 feet. From their heads the stalls are turned off to the right and left, opening a continuous face of coal. Mining is then commenced. The rails are laid and the haulage-ways

mm moved from the entries *ee*, where they are no longer of service. Work is simultaneously commenced in the haulage-ways *mm* to prepare for the continuance of the retreat from the gangways *dd* as soon as the operations have been completed on the face entries *ee*.

The main haulage-ways are in the return airways to the upcast and hoisting shaft *u*. Entries crossing the intake airways are carried over by air-bridges, Chapter XIV.

FIG. 4.—Longwall System of Mining Retreating.

Occasionally, when the roof is very weak, the gangways are driven singly and connected by cross-cuts 60 feet apart instead of being driven in pairs. This method, however, is productive of a greater amount of narrow work and a greater expense in ventilation.

The Combination Longwall System.—In Fig. 5 is illustrated a combination of the retreating and advancing systems. A portion of the bed above the bottom of the shaft is worked by the advancing system, while that below is driven by the retreating system. The seam is 16 feet thick, but does not prove very profit-

able because of a considerable expense incurred in driving the entries while planning the works. However, when properly executed it will relieve the roof, though there is always danger of a suspension of its subsidence which interferes with the "swing," and also in a collapse along an unexpected line.

The mining to the rise is conducted advancing in section A, while in section B the operations are in retreat.

FIG. 5.—A Combination of Retreating and Advancing Longwall Systems.

Comparative Merits of Longwall Retreating and Longwall Advancing.—In the matter of ventilation the retreating system is more perfect than the advancing system, as it can be regulated to better advantage. No danger need be apprehended from the evolution of gas from the waste. The cost of mining is the same in both systems, but the expense of making pack-walls and of opening and maintaining the roads is far greater than the increased cost of driving the narrow gangways through the solid coal. The large initial outlay in longwall retreating for the four pairs of main entries out to the extreme boundary of the property, and for laying tracks the entire length, and the long period of

waiting for the returns, constitute the chief obstacle to its more ready acceptance. But its aggregate advantages warrant its employment under all circumstances where the longwall system is at all applicable.

The Nottingham or Barry Modification is employed for mining splint coal of five feet in thickness with a strong roof and a clay floor. In this the gangways are 6 feet wide in the solid coal, with 20-foot pillars between them and the cross-entries 200 feet apart. From these are driven flats 120 feet apart and 200 feet long, the coal being attacked "long horn," Fig. 9, and to the rise. The cars travel along the working-face. Eight miners are employed on each stall of 120 feet, and produce large coal at low cost, with an excellent yield per acre.

In one mine, changing from room-and-pillar to longwall, the coal was opened by butt-entries on both sides of the gangways until their faces met, the stalls being 48 feet wide. The coal was strong and had a regular cleat overlaid by a slate top with a hard floor. It was blasted, not undercut, and 37 tons per keg of powder were obtained.

Pillar-and-room System.—The second system of mining coal consists in driving levels and opening long narrow rooms, from which the coal is extracted while advancing from the shaft, and in leaving pillars on either side of untouched coal which are subsequently recovered on the return. The rooms are frequently termed stalls and breasts, though the latter term is more correctly applied to the face of attack. When the rooms are very broad compared with their length, the term chambers is sometimes applied to designate them. The pillars left for support are also called ribs.

This system is a survival of antiquated methods and still prevails to a greater extent in the United States than elsewhere, rather because of local custom than from any special merit which it possesses. It is employed where the material for filling or gob is scarce. A great diversity exists in the various localities in the details of the plan. The height of the lifts, the direction of the rooms, their dimensions, the forming of pillars, the rob-

bing of pillars, the taking down of roof, etc., vary widely. Many of its modifications are due to physical conditions of the coal and roof, though the latter elements do not always receive the consideration due them. The facility with which the details of working may be varied to suit the irregularities of pitch and thickness, the rolls in the strata, etc., renders it especially acceptable in regions of geological disturbances.

The necessity for economy in the conservation of the mineral is not imposed upon the American operator. The question of output per acre is not of vital importance, and mining as a consequence is conducted here in rather a loose manner. Indeed, with the exception of culm-flushing there has been no improvement introduced in the anthracite exploitation during the past sixty years, though numerous mechanical improvements have been utilized to economize the handling and preparation of the coal.

The Typical Room-and-pillar System.—As stated in the previous chapter, the height of the lifts varies with the thickness and inclination of the seams. But this determined, the gangways are driven, after which the rooms are turned as fast as practicable to the rise or up the slope at an angle depending upon the pitch, as in Figs. 6 and 7, or on the mode of conveying the coal from the breasts to the gangways. The jaws or necks, *aa*, of the rooms *B* are about 6 feet wide for a distance equal to that allowed for the stump-pillars *A*, when they are suddenly enlarged to a working-face of 20 or 30 feet, as time and the condition of the roof will allow. The rooms are driven regular and uniform to a length of from eight to ten times their width, leaving a chain-pillar for support to the upper level. The rooms of each lift are opened progressively to the boundary as fast as circumstances will permit.

The cost of driving narrow entries and headings *aa* and *cc* is so high that modifications are introduced whereby the jaws *aa* are made as short as possible and widened out to the full face of the room, thus reducing the narrow work without weakening the stump-pillars. The pillars *P* left between the rooms are unbroken except for three or four small breakthroughs *dd*, through which

the return ventilating air-current passes from room to room on its way to the upcast.

Rock Pressure.—Assuming the specific gravity of rock to be

FIG. 6.—Rooms in a Seam inclined between 10° and 20°

2.6, then a block having a base of 1 square inch and a height of 12 inches will weigh 1 pound. A column 500 feet high will exert a pressure upon its base of 500 pounds per square inch. Each square foot of rock at that depth below the surface will

FIG. 7.—The Typical Pillar-and-room System.

receive a pressure of 72,000 pounds per square inch. The load which a pillar of coal will sustain is therefore proportional to its depth below the surface and the area of its base. A coal-pillar 25 feet wide and 200 feet long at a depth of 500 feet will receive a total pressure of 180,000 tons. To sustain this load without suffering injury it must have a strength at least equal to two or three times the load thus imposed upon it.

The Strength of Coal.—Experiments recently conducted upon the crushing strength of coal have obtained values ranging between 1600 pounds per square inch and 2100 pounds per square inch. The value for the ultimate strength of coal having been determined in a specific case, it is possible to determine the dimensions of a pillar required to sustain a given load transmitted to it from the roof.

Owing to the wide range of values obtained for the crushing strength of coal, it is advisable that each engineer determine the specific value for the coal of his locality, determining, thereupon, the dimensions of the pillars to be used in connection with this system of mining. Sample cubes should be selected and sawed from various portions of the seam, and their strength normal to the bedding ascertained by actual test. They should be procured from the breasts rather than the pillars.

Influence of Cleats upon the Resistance of Coal.—It was shown in the previous chapter that the direction of the cleavage of coal may determine the direction of driving the rooms and roads. The cleavage may also determine the resisting strength of the coal. In Fig. 8 is illustrated a block of coal with its horizontal bedding cleavage *aa*, and, extending from roof to floor, the vertical cleavage *bb*, as well as the butt-cleats *cc*, at right angles to both. In all coals this prismatic structure is pronounced in one or more directions, and along these planes a slip may occur before actual crushing ensues. The load which the coal can sustain is therefore less than its actual strength.

In Fig. 9 are shown arrows indicating the directions of attack upon the lines representing the plane of the main cleavage *bb*. The butt-cleat, line *cc*, presents usually a hackly appearance

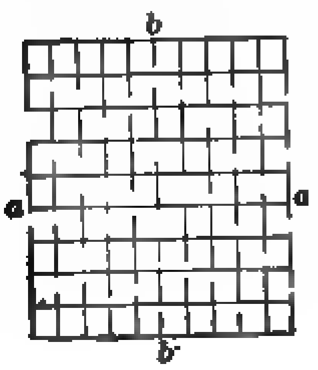
when exposed, and can be readily distinguished from the main cleats, which are perfectly smooth. Adhesion along the plane of the former is stronger, and hence, when conditions of haulage do

Fig. 8.—The Cleats in Coal.

Fig. 9.—Directions of Attack upon the Cleats.

not prevent, most mining is conducted in the direction represented by the arrow *c* at right angles to the main cleat, known as "face-on work"; an advance in the direction of the arrow *e* is "end-on work" mining; and in the direction of arrow *d*, "half end-on

Fig. 10.



The Directions of Cleavage.

work." The mining of coal at an angle greater than 45° and less than 90° is said to be "long horn," arrow *g*; and at an angle of less than 45°, arrow *f*, is called "short horn."

In Fig. 10 is illustrated the columnar structure of coal. Across the bedding *a* it is stronger than the cleat *b*, represented by the broken lines. In the horizontal position *a* is weaker than *b* (Fig. 11). In the inclined position (Fig. 12) neither cleavage plane is in its original position. The cleavages receive their respective components of the thrust *c*, the component *a* acting normal to the

bedding planes or across the main cleat, while the component e is parallel to the main cleat. The amount of thrust falling normally upon the bedding plane is less as the inclination increases.

Size of Coal-pillars.—The width of the coal-pillar depends upon the strength of the coal and the depth of the rock, rather than on the strength of the roof. When the rooms are open each pillar must support not only the rock immediately over it, but also sustain the pressure transmitted to it from the roof over the room. A rigid and firm roof will transmit its entire pressure equally to the sides. Hence the coal, at a depth of 500 feet, in the pillars 25 feet wide and 200 feet long between rooms 20 feet wide, will receive a pressure equal to the weight of a mass 35 feet wide, 200 feet long, and 500 feet high, or about 252,000 tons. This corresponds to 50.4 tons per square foot. So long as this is less than the breaking strength of the coal the pillar will remain intact. If the crushing strength be 2000 pounds per square inch, the stress upon the coal is less than half its ultimate strength. The margin or factor of safety is 2.8. This factor will be further reduced if the planes of main cleavage are pronounced and weak.

At a depth of 1400 feet there remains no margin for safety, the limit of the coal resistance being reached with its relation of room to pillar. Below this depth the pillars must be wider or the rooms narrower if the system is to be pursued. When a roof is brittle it will break without transmitting its pressure to the pillar. The pillar is relieved of some pressure, in which event its width is not necessarily increased. A soft floor requires a broad pillar to distribute the thrust over a large area of the floor. Otherwise a heaving of the floor ensues, which disturbs the tracks and "creeps" over the territory, closing rooms and airways.

The Relative Dimensions of Pillars and Rooms.—The width of the pillars depends upon the character and length of service they are to perform. Safety demands large pillars, but inasmuch as only a portion of their coal contents is recoverable during the second working, a very large percentage of the total in the tract is lost. A rapid production demands wide rooms which can be mined by machinery and exhausted quickly, but they require a

good roof and, in turn, broad pillars to support them. The relation between the two must be determined by local conditions, observing the principles mentioned above. Narrower pillars with wide rooms are decidedly wasteful of coal, and in a five-foot seam with ribs of 12 feet and rooms 36 feet, none of the pillar coal is recovered, whereas in a 6-foot seam with 12-foot rooms and 28-foot pillars nearly 90 per cent of the coal contents are sold, particularly if panel pillars are also employed in sections of the mine, as illustrated in Fig. 17. The usual proportion of pillar to room is about 3 to 2 for the average depth of mining as prevailing at present in America.

As an illustration, it will be found on calculation that in a mine opened with rooms of 12 feet width between pillars of 18 feet a greater concentration of labor is obtained, and there are positive sources of economy over that possible if the rooms and pillars alike are 20 feet wide. Along a mile of gangway 176 rooms can be opened in the former and but 132 rooms in the latter design. Though the aggregate breast length is less, there will be more rooms for simultaneous work. Or, for an equal number of working-faces, there will be a concentration of work by the narrower pillars and rooms, which saves 3520 feet of track for the 132 rooms; haulage will be less; and 44 cars can be obtained in the same time from three quarters the distance of that when the rooms are spaced 40 feet apart. In robbing the pillars more coal will also be recovered from the relatively thicker pillars.

The width of pillars is increased and that of the rooms decreased as the depth of the vein is great. The limit to an economical application of this system is reached when the rooms are reduced to an unprofitable portion of the deposit. At a depth of 700 feet they contain less than 30 per cent of the total contents of the vein. At 1600 feet depth the pillars leave only 25 per cent of the coal for the rooms; while at 2000 feet the rooms are one fourth as wide as the pillars, and are capable of producing not more than 15 per cent of the total coal.

The general impression that coal-pillars and beds of steep pitch are less strong than those in flat beds is untrue unless a

deformation of the strata has produced a fissility in it. Their resistances are equally great and their life as great, being diminished only by a protracted period of pressure and by alterations of temperature and humidity which injure the integrity of the pillar. This is independent of the inclination of the bed.

There is an economical limit to the narrowness of the rooms, based not upon physical laws but upon an artificial law by which a working-face of less than 15 feet is considered in many coal-fields as "narrow work," making the cost of driving such rooms greater per foot of length than that of wider rooms. Moreover, narrow rooms do not admit of an improvement of machine-cutters or large gangs of loaders, and therefore increase the cost of coal.

The Mode of Delivering the Product from the Face to the Entry.—The most important problem in mining is the method of transporting the mineral from the working-face to the entry or gangway. It usually determines the plan of operations. When the vein is vertical or nearly so the mineral may be dropped through mill-holes or chutes to the cars in the level below (Fig. 87), or lowered by means of a self-acting incline (Fig. 115). In the anthracite region the working-face is termed a breast. Here the broken coal is allowed to accumulate in the batteries (Fig. 13), which are loaded at the coal face and more or less completely emptied at the foot from a platform, the flow of mineral being controlled by a gate. So long as the pitch exceeds the natural slope of broken rock, 40°, the coal will roll or slide on the rock bottom without the necessity for lining the latter. The sides may be built of plank spiked to the props, reaching diagonally from the floor to the roof, and the lower portion closed by a gate or door with a platform. By keeping the chute or battery full of coal and emptying only the excess through the gate, the loss from attrition will be slight. This, of course, is not possible with bituminous coal. At a grade flatter than 40° the chute must be floored with wood or iron, but below 18° soft coal will neither slide nor roll. If the working-room is large, the breasts wide, or the vein thick, the chutes are carried up with

the work, one on each side of the waste filling. In small rooms the chutes are built centrally with the manway compartment alongside of them (Fig. 13).

Beds having an inclination between 18° and 6° from the horizontal are the most difficult to provide with an economical method of transportation. They are too flat for self-acting systems and too steep for any simple plan of haulage. An endless-ropeway plan for each room is too complex, and any scheme involving shovelling would be both expensive and injurious to the coal.

FIG. 13.—Battery Mining of Thick Steep Seams.

In such event buggy-roads are resorted to for the rooms (Fig. 6). The track is laid in sections directly upon the floor, or upon the trestle raised at the lower end to moderate the grade, according to the headroom that may be afforded and the pitch of the seam. Another method consists in having jig-planes where the loaded car in its descent pulls up a weight which, on the next down trip, raises the empty car back to the face.

Breasts driven up a grade of less than 10° are said to be flat, and the accessory haulage from them presents no special difficulty. The rooms may be driven directly up the rise for the small angles, or, when the pitch reaches the limit, are laid so as to secure a satisfactory grade. In such cases mules may be utilized for the power, or in exceptional cases man-power is employed.

Rooms in beds as flat as 3° are driven directly up on the pitch, employing mules or men for transportation of the cars.

The rooms are rarely worked downward in the dip, unless the pitch be very slight. For a short distance an undulating seam may require this modification.

Robbing the Pillars.—The rooms, having been carried to their length, are then turned and "butted off," the pillar at the top driving along the under side of the chain-pillar. The pillars are then attacked and their contents removed as soon as possible, before their coal contents have suffered in quality from pressure. The operation is performed as rapidly as possible for safety, and the pillars are taken in successive retreating. Their sides are scaled off as much as can be secured before the pillar crushes. The heading may be driven off the centre of the pillar and some of the middle portion of the coal obtained, leaving two thin supports for the roof; or the pillar may be worked over its entire width from the chain-pillar toward the gangway.

The stump- and the chain-pillars are not disturbed until the lift which they support is to be abandoned, when their contents are promptly extracted and the roof allowed to cave.

Fig. 14 is the plan of a flat seam operated as described above, the two sections of he property being separated by the double broken lines. From the outcrop at *a* and *b* the main and return headings are carried 60 feet apart. The cross-entries are 360 feet apart and the rooms 300 feet long. The outer irregular line is the outcrop. The fan is located at *c*. The break-throughs *d* are midway in the rooms and on one line to serve as an auxiliary haulage-way during the robbing of the pillars. The jaws of the rooms are 10 feet wide and 15 to 20 feet long. The room is widened in a direction away from the outcrop. Pillar-robbing begins at the outcrop. This process, being slow and delayed till all rooms are mined, results in an excessive waste of coal. The speedy returns, good ventilation, and safe work commend it.

Comparison of Longwall Advancing with Pillar-and room System.—The relative advantages and disadvantages of these

two systems are easily stated. In the former the expense of maintaining roadways is higher, the number of accidents greater, the product of round coal much larger, the product

FIG. 14.—A Double-entry System in a Flat Seam.

per acre higher, the ventilation of the face simpler, the amount of narrow work less, and the consumption of powder less. On the other hand, the latter system is more advantageously employed where faults, horsebacks, and dikes may be encountered,

if the upper strata are wet, if the boundary is at a great distance, if the mines are shallow, or if the surface land is very valuable.

Modified Pillar-and-room Systems.—In some fields the cross-entries are driven 360 feet apart and the rooms are turned from each entry, making them about 180 feet long; in such event the cross-entries are carried up the slope independent of the direction of the cleavage. The only feature commending this plan is the reduction in the amount of narrow work and the yardage thus saved.

FIG. 15.—Connellsville Method.

A modified double-entry system employed in the Connellsville coke district drives gangways 1000 feet apart (Fig. 15), and four pairs of headings from them at distances of 400 feet. The rooms are turned when the latter entries have gone 1040 feet and are butted off at the chain-pillars, after which the pillars are robbed. The coal being soft, the headings and the jaws of the

rooms are driven 8 feet wide, the pillars 30 feet, and the stump-pillars between gangways and headings 52 feet.

The double-room modification giving satisfaction in the Southern States for seams of 5 feet thick with weak roof is shown in Fig. 16. The main and cross entries are driven in pairs, and later the rooms are started right and left 42 feet wide with pillars of 36 feet; their length is 180 feet, but they are not opened fully until the cross-entries have reached their boundaries. Props

FIG. 16.—The Double-room System.

are set along the inner side of each roadway for the gob, and waste is thrown into the centre of the room. When they have reached their extreme length the rooms are widened and one half of the pillar on the floor withdrawn retreating. When the succeeding rooms have reached their limit each is widened in turn, the remaining half pillar on one side and one half the adjoining pillar on the other side are cut away. Props set along on either side of each roadway are removed or broken during the retreat, in order to facilitate the uniform fall of the roof.

Mining Steep Thick Seams of Coal.—The coal in a steep bed may be attacked in steps proceeding upward to the top of the given lift, the excavated space being filled with rock obtained from elsewhere. In such an event provision must be made for planked chutes to deliver the coal without loss or admixture with the waste rock. The broken coal may be allowed to accumulate in the excavated spaces as a partial support to the roof, and also for the men while mining the coal overhead. If the rooms are not too wide, the coal will not suffer in quality by the time the room is exhausted. The other system of mining thick seams of coal contemplates its removal over its entire length, proceeding downward from the top of the lift or from the superficial covering of the seam and allowing the material above to cave upon the work as it progresses downward. These methods resemble somewhat the two longwall systems as applied to the flat thin beds.

The famous mammoth bed of the anthracite fields of Pennsylvania presents the greatest difficulties to the operator. It varies in pitch between wide limits and within a small area attains to a thickness of 110 feet.

A Room System for Gaseous Mines.—In Fig. 17 is shown the Brown panel system employed in thick seams of coal where there is considerable gas and the roof poor. At distances along the gangway of 180 feet are turned double rooms up on the pitch. These are 24 feet wide, separated by a pillar of 15 feet in thickness, which latter is withdrawn as soon as the rooms have reached the upper airway. Midway between each pair of rooms is driven a central heading 12 feet wide to the full length as a travelling road and airway, as well as a coal-chute. From this, to right and left at intervals of 30 feet, are turned similar headings carried through the coal until the double rooms are reached. The coal is mined from the rooms toward the central heading as indicated in the figure, and in such order as to leave flanking pillars as shown. Small cars convey the coal to the chute. A heavy barrier pillar separates each pair of rooms, and is untouched until the boundary of the property has been reached. The sys-

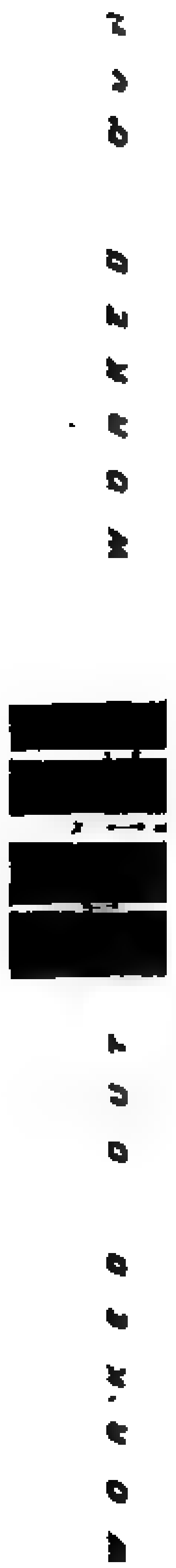


FIG. 17.—The Brown Panel System for Gaseous Mines.

tem affords safety to the men, but is expensive because of the large amount of narrow work performed before much coal is produced.

The County of Durham System is a combination of the panel and the pillar-and-room methods. The breasts with their pillars are laid in groups of eight or ten from the gangway, each section or group being mined separately and systematically toward the boundary. Between the sections are left barrier-blocks of coal 150 feet thick, unbroken except for airways. These give complete isolation to the section and also localize whatever movement may occur in the falling of the roof. As the coal in these barrier-pillars suffers in quality from the crush of the roof, their location is selected where the coal is of poor quality. They are not touched until all of the adjacent sections have been exhausted. This method is safer than the pillar or room in gaseous mines, and more economical than the panel method.

Anthracite Mines.—The mining of anthracite presents conditions that are not usual in other fields. The contortions and folds to which the strata have been subject during the processes of mountain-making have developed so many eccentricities of pitch that it is difficult to plan the works a great distance in advance. For example, in a shaft-mine in which the workings may have continued horizontally for some time, the coal might suddenly rise or dip and change the entire topography, and hence necessitate an entire revision of the system. The system generally adopted in the anthracite region is the pillar-and-room on the double-entry plan. Levels are driven each way from the shafts sunk to intersect one or more beds. The coal is invariably worked from the lower toward the higher level, and the pillars are shot through, leaving chain-pillars as a solid support for the level above. The width of the rooms and of the pillars is not materially affected by the pitch of the beds, though, at times, elaborate timbering may be necessary to safeguard the men and prevent the coal from running into the gangways.

Rock-chute Mining.—In the Wyoming and Lackawanna valleys, where several coal-seams are worked simultaneously,

special care must be exercised that the rooms in the different seams are laid off in such manner as to leave their pillars in the same vertical line. When the beds are thick and separated by partings of slate of but a few feet, or the several beds are thin and separated by a few yards of rock, they are mined simultaneously, all being connected by a chute built from the lower to the upper bed for delivery of the product to the lower seam.

Systems of Mining the Mammoth Coal-seam of Pennsylvania.— The system followed at Hazelton, Pennsylvania, in mining the Mammoth bed at a point where it is 40 feet thick, with an inclination at times of as much as 60° to the horizontal, is shown in Fig. 13. This is known as a single-chute breast method. The plan shows the gangway *a* heavily timbered through the coal, and a chute *b* leading from the battery platform to the gangway. The battery *c* is made of open timbers hitched into the floor and roof, backed by horizontal timbers. In it is a door of planks capable of being removed when desired for loading the cars. A chute in the middle of the room receives the coal, and manways, *dd*, on either side, made by leaning posts diagonally from the floor to the rib of coal, provide air and entry for the men. These manways are often termed "juggler-ways." A breakthrough, *e*, is made between the rooms for ventilation, and the main airway, *f*, is driven over the gangway, as usual, near the roof. Every third or fourth room is connected with the intake and return airway, and in extreme cases every two rooms are also connected. The amount of coal which is withdrawn from the battery is only sufficient to relieve the battery of the excessive volume, sufficient being allowed to remain for the miners to stand upon while engaged in work.

A modification of this single-chute breast method consists in providing each room with two batteries, the manways *dd* serving as chutes, the space between them being used for refuse. This is the plan employed when a large amount of bony coal and slate is present in the seam.

Propping through a Weak Floor and Roof.—At the Richard mine, where both the floor and the roof are weak, a variation, as in

Fig. 18, has been introduced. The props are placed 6 feet apart, with their length such as to drive them through the weak shale to the solid rock beneath the floor and likewise above the roof slate. By this plan an ample support is given while the pillars are being extracted.

FIG. 18.—Variation of System for Pillar-robbing.

The Room System with Caving.—In the North Staffordshire mines is a system similar to the one mentioned above, in which the gob follows downward with the mining and requires no attention. This has also been employed in California, and resembles closely the caving system as practised in the iron-ore mines of Lake Superior, as described later on in this chapter.

A Room System with Filling.—At the Shamrock colliery, Westphalia, is practised another system of mining coal at 45°, the bed having a thickness of seven and one half feet. Here the bed is divided into blocks 2000 feet long on the strike and 600 feet on the pitch. Each Block is subdivided into three levels (*aa*, Fig. 19), 200 feet apart, through which is driven the central heading *a* for

ventilation. On each side of the pillars are self-acting inclines, or coal-chutes, *ee*.

This method contemplates the filling of the excavation made by the removal of the coal by such material as can be obtained at the surface. The central headings *aa* serve for lowering the material used for storage. At the point *c* is a small shaft in which is accumulated the filling material and such rock as is obtained in taking up the floors of the haulage-ways. From this point is

FIG. 19.—A Room-and-pillar System adopted in Westphalia.

obtained the material which is delivered to the rooms through the central heading *aa* when needed.

The plan of the mine provides three working places on each side of the central heading, and during operations three of them are being filled, while those on the opposite side are in the process of excavation. This alternation of operations maintains regular pressure of the roof.

The coal is removed by a process of overhead stoping (Fig. 27), delivered to the levels *a*, and thence by the inclines *e* to the main levels. On either side of the central heading is left the stump-pillar, which is eventually withdrawn when the level is to be

abandoned. Its lower end is very carefully blocked to prevent the leakage of air from the ventilating current.

Mining Thick, Steep Coal-seams with Rooms.—The method of mining shown in Fig. 20 is described in the Coal- and Metal-Miners' Pocket-Book as having originated in California. The coal-seam is 7 feet thick and has an inclination of 60°. The gangway *a* is driven along the strike 40 feet from the airway *b*; breakthroughs, *cc*, at distances of 30 feet connect the two, acting first as airways and

FIG. 20.—A Californian Variation of the Pillar-and-room System.

later as mill-holes or chutes down which to lower the coal. The door or gate at the foot of each chute permits the coal to be loaded directly into the cars on the gangway. Counter airway-chutes *dd* are driven over 30 feet apart at an angle of 35° for delivering the coal to the chutes, *cc*, by gravity. Numerous breakthroughs, 40 feet apart, divide the coal and the lift between each two pairs of levels into pillars. When a panel of five or more chutes has been driven the pillars are robbed on the retreat from one corner at the top. This is possible only in comparatively strong coal without partings.

Square Work.—Square work is a modification of the pillar-and-room system, in which very thick beds, not only of coal but also of salt gypsum, puzzolana, etc., may be mined. Rooms

are opened from the gangway 150 feet square, with pillars 30 feet thick between them.

Two sets of cross-galleries 20 feet wide are driven through the rooms, leaving pillars about 25 feet square for the support of the roof. They are driven as high as the vein-matter will allow. If the roof or floor is not firm, a layer of mineral may be left for greater security. The objection to the method is in the difficulty of obtaining systematic ventilation. Moreover, very little of the pillars is recovered, and the system involves great risk to life, since the roof is usually too far for the eye to detect any threatening disaster.

Flushing.—About 15 per cent of the coal obtained from the anthracite mines is of such a nature that it requires special treatment, which may not warrant the expense of attempting to recover it.

Moreover, in breaking and sizing the coal for the market a large quantity of fine coal is produced and slate sorted which were formerly dumped and accumulated in large piles, encumbering the surface and increasing the pressure below. These culm piles are now being utilized for the filling of the excavations by a process known as flushing. In Scranton, Pa., a stream of water is discharged against the culm piles, washing the material into a pipe-line and down holes driven from the surface to the underground workings. The mouth of the rooms being boarded up at the gangway in such a manner as to drain off the water without allowing any culm to escape, the room becomes filled with wet culm. The culm packs quickly and solidly and offers a degree of solidity almost equal to that of the untouched coal in the pillar. The pillars are then attacked on either side with safety and their contents almost totally recovered. The roof is then allowed to close in the space and rest on the floor and culm.

Metal-mining.—The location of the point of attack of a metal-mine involves considerations which were discussed in Chapter II. The comparative merits of the shaft driven on the wall side of the vein, and of that intersecting the vein, as illustrated in

Figs. 22 and 23, were likewise reviewed. The determination of the system of mining depends largely on the capital available and the value of the deposit. In precious-metal mining

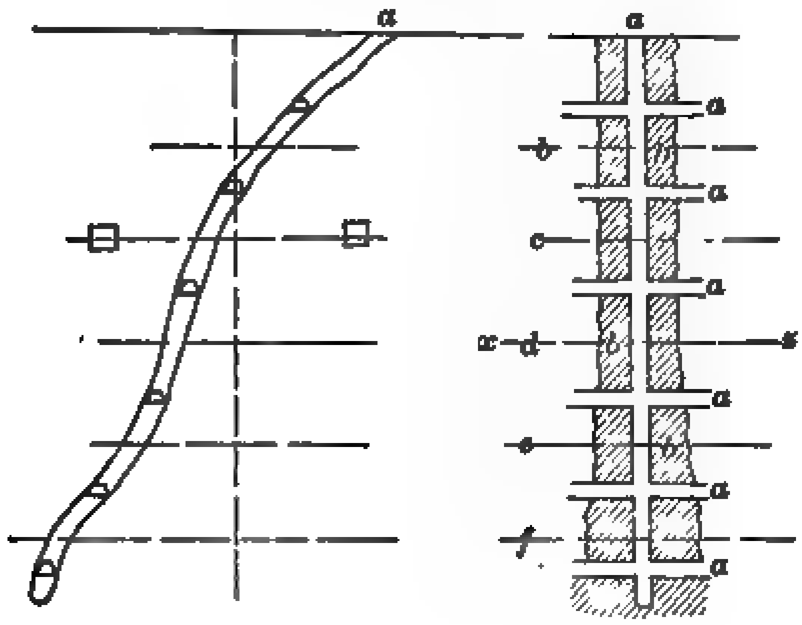


Fig. 21.—Following Vein by Incline.

the mineral cannot be worked from level to level as rapidly as the levels are pushed forward, for it is difficult to estimate in advance the quantity of "pay ore." Hence for economical

Fig. 22. Fig. 23.

The Two Positions for a Shaft Relative to the Vein.

work only sufficient ore is extracted to pay the expense of operation until the limit of the property has been reached. This assumes on the part of the operators a degree of patience which is not usually found in speculative districts. Under ordi-

nary circumstances it is desirable or may be necessary to obtain immediate returns. In Fig. 24 is shown the method pursued in blocking out the mine. It represents a longitudinal view of the vein. Through the vein-rock extend streaks of ore of high grade which are neither uniform in dimensions nor in pitch. These may be encountered at any place in the vein-matter. In such cases the risers *a* are driven, or winzers *b* are sunk, in such of them as promise profit. If the ore is uniform in grade and value, the mill-holes are spaced uniformly. These serve to develop the property, enabling the future to be calculated with

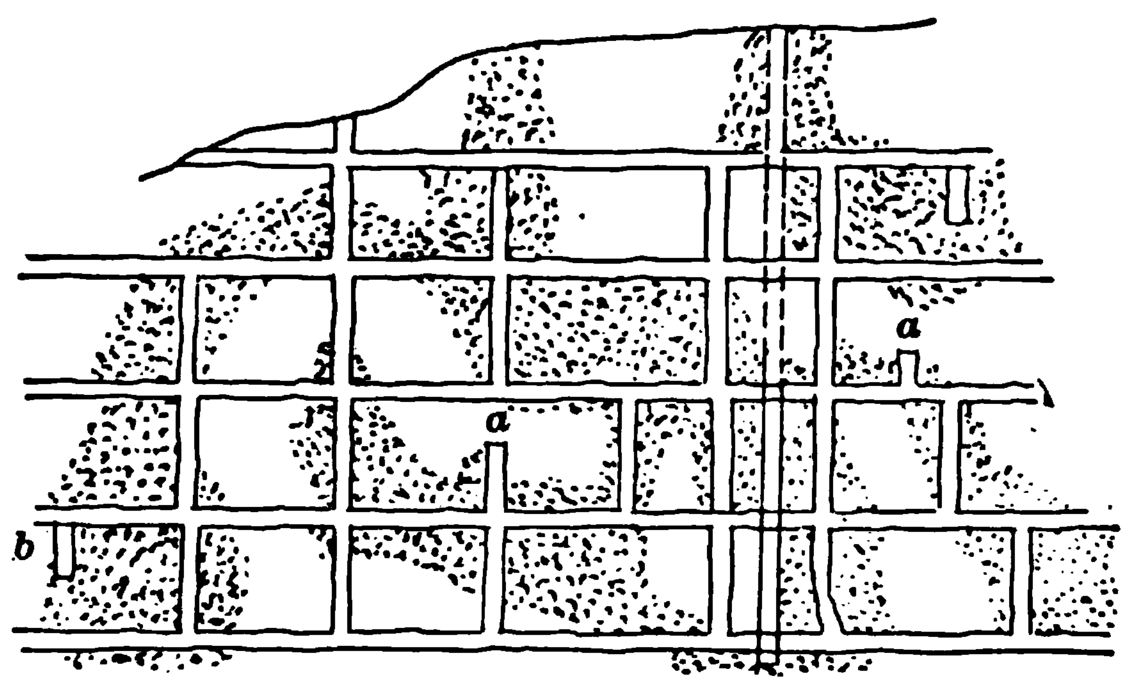


FIG. 24.—A Vein Blocked Out, showing Ore-shoots.

some degree of certainty. The mill-holes serve for ventilation as well as ore-chutes. Risers are cheaper to open than winzes, and are preferred in wet mines, though they are at a disadvantage in the hot mines.

Systems of Mining Metalliferous Veins.—The precious metals occur usually in thin veins of hard rock, and constitute a small portion of the deposit, and the systems adopted for their extraction are known as *underhand* or *overhand stoping*.

In overhand stoping the mineral is recovered by upward workings in a series of steps, the barren rock being stowed away in the lower portion of the lift. In underhand stoping the ore is mined out in steps working downward, while the waste rock is disposed of on platforms built to receive it.

For thick vertical deposits of iron, copper, lead, zinc, or other ores, three systems are in vogue, varying with the strength of the ore. The *square-set* and *filling systems* are used in large deposits and veins for removing the entire contents, whether they be friable or strong. The *caving system* is designed for ore in large masses friable enough to break and fall when undermined. The order of proceeding in all these systems is upward. A descending system is employed when a heavy cover can be utilized as a flexible roof, which is allowed to cave with the downward progress of the excavation.

The Stoping Systems of Mining.—After the vein has been blocked out, then such blocks as are deemed workable are mined by one of two methods, *overhand* or *underhand.* In the first method the miner picks or shoots down the ore in front of and above him, advancing the breast as high as he can reach and as wide as the vein and as far as the limit of the block. In underhand work the miner removes the mineral below him and advances the breasts longitudinally, with the load taking them successively downward. These two methods are applicable to steep veins if the overhand working is quickly adjusted in the thick flat beds. In the former method they are merely divided from the falling rock, as gravity facilitates all operations of breaking down the vein-matter; this is a much quicker, cheaper, and less arduous method than that of underhand, as the miner is compelled to raise the mineral before the next lower can be attacked.

The Underhand-stoping System.—This is a method of limited application. When the vein has been blocked out and the several blocks on any given level are attacked, they are worked away in horizontal slices, beginning at the winzes and extending half way on either side to the adjoining winzes. The miner standing on the floor of the level (Fig. 25) removes the vein-matter for a depth of nearly 6 feet, sorts out the ore, and throws upon the platform behind him the waste. When these slices have been carried as far as designed, the next lower is attacked, beginning at the winzes, its ore being raised to the proper level and its gangue thrown upon the platform behind the miner. The platforms are constructed

as rapidly as the miner advances, there being one for each horizontal slice. If the entire contents of the vein from wall to wall are composed of pay dirt, the timbering may be dispensed with.

It frequently happens that instead of a single gang operating in a given block on one slice at a time, as many gangs are disposed in the work as there are slices, the uppermost gang preceding the one below, in each case by an amount sufficient to secure for him the advantage of the firm footing on the ore. Thus the mine progresses, as illustrated in Fig. 25, in which the block resembles a set of steps downward from the level above, whence the

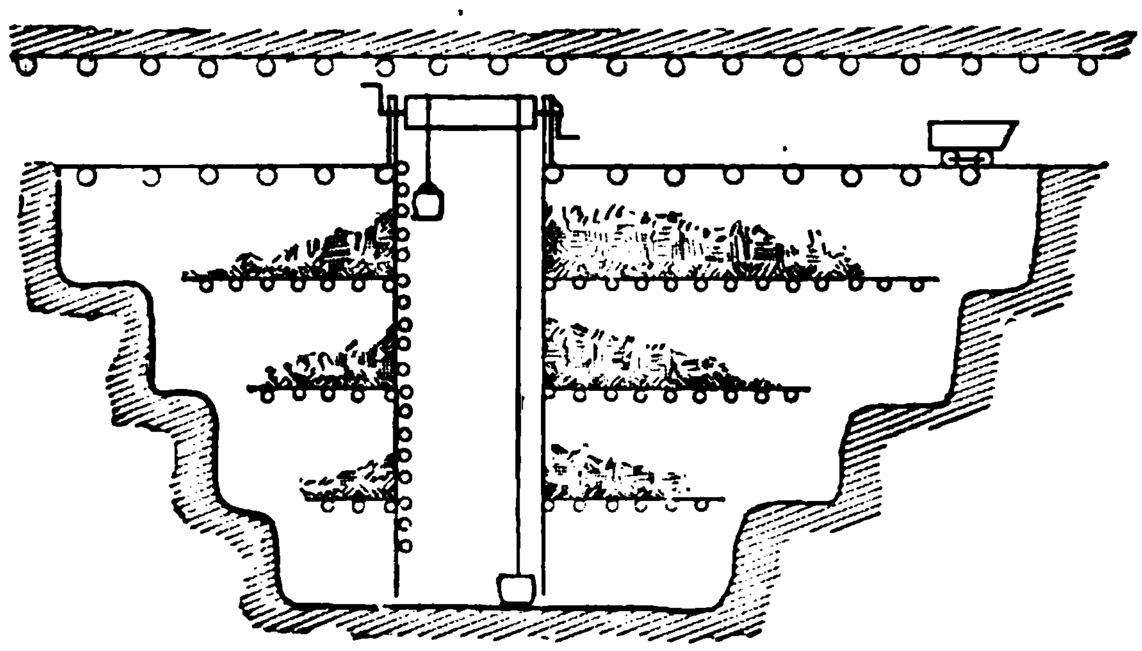


FIG. 25.—Underhand Stopes.

name underhand stoping. Occasionally the winze is sunk to the lower level at once, and each step is advanced as rapidly as possible, the mineral from all the slices or steps being delivered downward to the level below, instead of requiring to be raised.

This system of underhand stoping is economical and has many advantages as compared with the overhand system to be described later. Timber is used only for the support of the fragments of rock and for building platforms. The ventilation is good, and better drilling can be conducted by the men in underhand than in overhand.

The iron-ore deposits of the Eastern States are worked downward, the vein being entirely removed except for the occasional pillars left to support the walls. The deposits vary in thickness, the average being probably 12 feet, and in pitch between 15°

and 70°. In Fig. 26 the shaft pillars are at *aa*; those at *bb* support the roof; at *cc*, the surface, and at *gg*, the track in the upper level. At *dd* is the untouched ore. The latter is broken down from the stopes, *ee*, with air-drills and loaded into cars below. The levels are 60 feet apart. Each stope can furnish

FIG. 26.—The Underhand Stoping of Iron Ore.

about 50 tons in a single shift. The hoisting-car track follows the foot-wall except where it is too irregular, when the hollows are bridged over or the projections cut away, as at *b*, to obtain a regular slope.

The Overhand Stoping.—The ore in this case is mined from below upwards, as shown in Figs. 27 and 28. The attack upon

FIG. 27.—A Double-wing Overhead Stope.

the blocks of mineral begins at some point on the upraise, the miners standing on the caps of the timbering in the gallery, as shown on the extreme right or left in Fig. 27. The miners remove a

horizontal slice for a height of about 6 feet, as far as desirable, sorting out the ore and delivering it to the mill-hole, where the chute discharges it into cars below. Meanwhile the waste rock is piled on the gangway timbers behind the men. The next upper slice follows, and is broken away in the same manner from the mill-hole, the miners then standing on the waste rock from the previous slice. If the vein can supply no waste rock, a temporary

FIG. 28.—Longitudinal Section of Stope.

staging is placed conveniently for mining as is shown near the top of the room in Fig. 28. The room in this case presents an appearance of inverted steps, whence originates the name. In the double wing stopework advances simultaneously from both sides of the mill-hole, as in Fig. 28.

In shooting down the ore, sheet iron or boards are laid on

the floor to receive the mineral and prevent the pulverized and friable ore from being lost in among the waste.

Comparison of Underhand with Overhand Systems.—The comparative merits of these systems depend largely upon the amount and cost of timbering. From this point of view the overhand has the preference, as requiring little or no timbering for the reception of the waste. The underhand requires a platform for each step in each room. If the vein is entirely salable, there is then no waste, and very little permanent timbering will be needed for underhand stoping. The underhand system is accounted best for valuable ores because all rock must be handled, and there is no occasion for any being lost. On the other hand, as the ore is being trodden upon continually, it cannot be used for brittle minerals. Overhand work affords some advantages in regard to breaking down mineral. In a wet mine the underhand stoping presents objections unless the mill-holes and winzes have been sunk to the lower level. The overhand method is practicable in a wider vein than is the underhand, which is limited to a width of vein by the length and size of the stulls which are available.

The Square-set System. — When the vein of metalliferous mineral is wider than 8 feet, the underhand-stoping method used to introduce the timbers to reach from one wall to the other must be exceptionally long and correspondingly stout, hence is very difficult to handle with the conveniences available underground. The method then employed may consist in the substitution of a set of frame-timbers (Fig. 258). This method of square-set timbering is applicable to either hard or soft ore. It is equally applicable to flat as to steep deposits, and succeeds just as well with soft ore as with firm ore.

The vein is blocked out in the usual manner into lifts of 60 feet each by gangways or levels running longitudinally in the vein, if its contents are of hard rock, or in the country rock if soft.

Along the gangways are alternate rooms and pillars, each of the same width, from 20 to 40 feet long, and as wide as the deposit. The breast of each room is divided into working-faces of about

7 feet wide. The erection of the square set of timbering begins as soon as possible, props and wedges being placed between the timbers of the set and the roof or side rock to prevent fall. With the advance of the room additional timbers are laid in the frame on the floor, at the side, in front and above; thus, as the room is being excavated to its limit, the framing assumes a form similar to that in Fig. 258.

The square setting of this type is employed in mining chambers or deposits when the ore is of a comparatively uniform nature. It is most economical, for the steps can be worked on all four sides. When the first sets are fully under way for a considerable area, the first floor is stoped out and sets placed directly over the sill-sets, care being taken to be at least two sets behind those on the first floor, in order to keep sufficient ore on the floor to work with while the sets are being placed in position.

The operations of mining may be confined to one level only, or they may be prosecuted simultaneously from several levels working across the vein and upward in each lift. The mining proceeds by slices, ascending to the upper level, as in the systems of stoping, the timbers of the respective sets being placed in vertical tiers. The mining is aiso conducted in an order resembling longwall, in which the entire face is attacked and the timbering placed at once, the progress then being in every direction except downward. This modification is favored where the mineral is of low value and very soft.

When the rooms on a level have been exhausted and the timbers carried to the upper level, work begins upon the intermediate pillars in the same manner as practised in the rooms.

In Fig. 29 is illustrated the method of stoping with square sets where the work is prosecuted from the hanging-wall side of the deposit and carried across it. This plan assumes the hanging-wall to be heavy and liable to cave or the ore soft. The construction of the square sets, *b*, is then conducted as rapidly as sufficient space is excavated to permit of it. The set *c* is next provided for and erected, after which the ground above them is stoped out for the high set *d* and the flat set *g*. The set *e* is next erected, as shown

at *f*, and the sets built up in succession until the hanging-wall is reached.

To avoid any risk of movement, care must be exercised that the posts be perfectly in the line of pressures and are securely

FIG. 29.—The Square-set System, working from the Hanging-wall.

blocked by wedges at the points of contact with the hanging-wall. Where there is any pressure the sets are reinforced by braces, as at *h*. Occasionally ladders *ii* are carried up for the men, and chutes *j* arranged for the delivery of the ore into the bin. All ore is removed from the vein. The support for the roof depends upon the timbering alone and upon its security. Care must therefore be taken to prevent caves.

In Fig. 30 is an illustration of square-set timbering beginning at the foot-wall. The rock is first cut for the sills *a*, which are

placed in position, and the sills *b*, of extra lengths to secure a firm base. Sometimes they are of especial lengths, as at *d*. Hitches are cut directly under the posts of the set above and the short posts *c*, the caps then having different lengths to simplify the framing.

When the deposit contains considerable barren material which can be employed as waste and thrown into the spaces

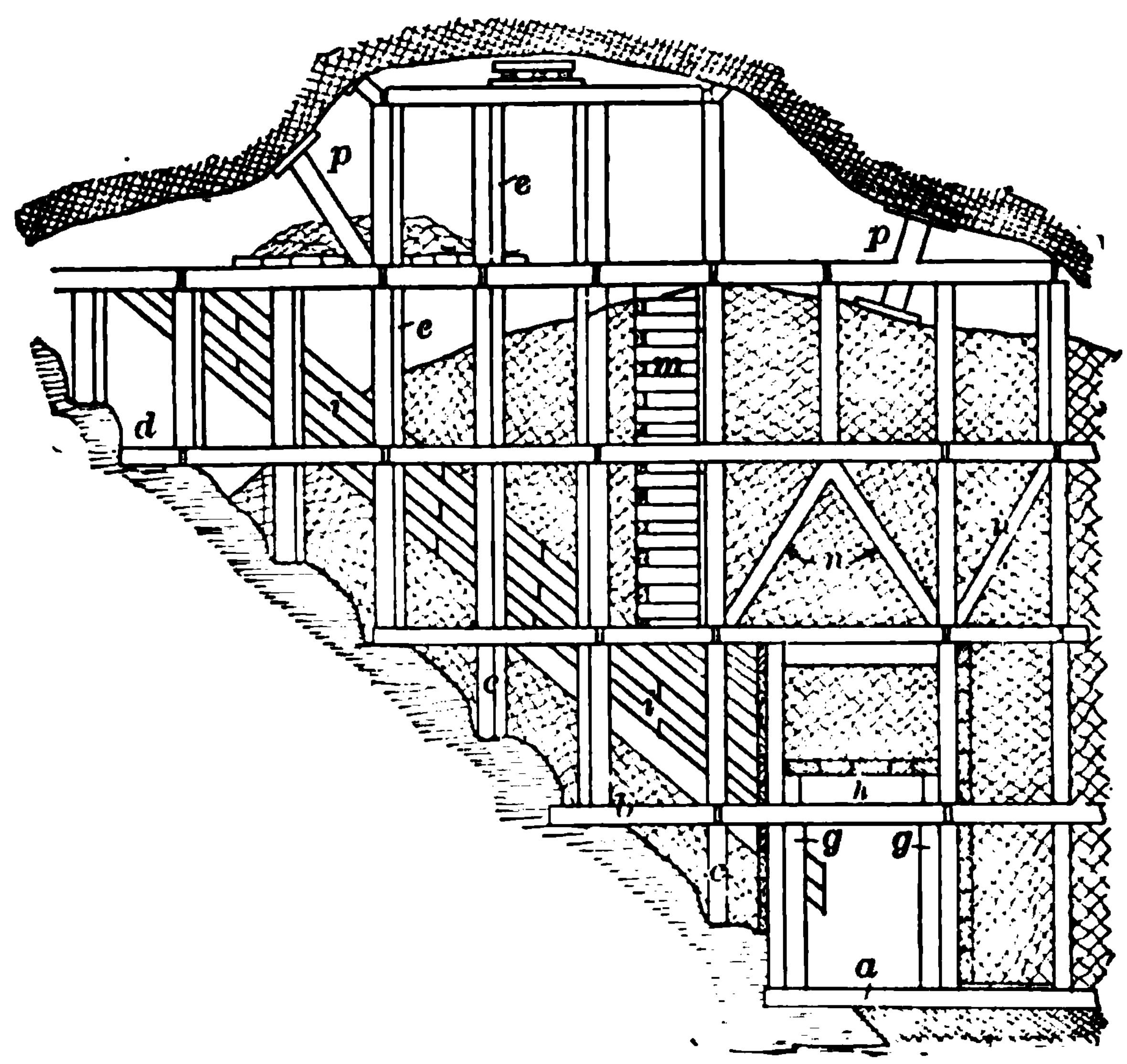


FIG. 30.—The Square-set System, working from the Foot-wall.

between the sets to accumulate and eventually assist the timbers in sustaining the pressure, the sets at the level are reinforced by auxiliary timbers *g* and *h*, and also lagged to keep the level open. The mill-hole *m* is surrounded by a gob and serves as an airway. At *p* are props used to temporarily support the ground and to guard against accident during the operations of stoping, and the single braces are introduced to prevent deformation of the frames.

In the Lake Superior region several of the iron-mines are worked in this manner, the lower sets being subsequently filled with rock or sand lowered into the mine from above.

Top-slicing and Caving.—There is a system of mining which has been successful in wide deposits where the cover can be induced to break and follow the excavation from a higher level to a lower one. The mineral is removed from the top of the deposit in horizontal slices 8 feet wide and 6 feet high, made by a series of horizontal drivages timbered with the three-stick sets (Chapter III, Part II) as the work progresses. On one side is the solid ore, on the other is refuse of previous slices which had caved when the timbers were removed.

The shaft is sunk in the country rock, and from it, at levels of 60 feet each, are driven cross-cuts to the deposits. The first cross-cut is extended from one wall to the other, and from the centre of it are driven levels at right angles to the outer boundary lines, at intervals of 100 feet; on this level, risers are carried up nearly to the cover, from which are run cross-drifts until the side walls of the deposits have been reached. The horizontal slice is then removed at the top parallel to the wall and the space timbered. Its length on either side of the cross-cut does not exceed 50 feet.

In Fig. 31 is shown this system, in which *a* is the main level,

FIG. 31.—A System of Top-slicing the Ore.

b the riser, *d* the cross-cut on top of the ore, and *e* the cave following the slicing. A riser is made 6 or 8 feet and divided into two compartments, one as a chute and the other for a ladderway.

No effort is made to support the roof, and the cover is there-

fore allowed to cave, the caving taking place as the men retreat from the end of the walls toward the risers, and from the ends toward the cross-cuts, and from walls toward the risers.

When the excavation has been carried 50 feet from the cross-cut, the timbers are withdrawn, or cut or blown down, as the case may be, in order to permit the roof to cover as soon as the work is completed. Prior to removing the timbers, however, there are placed on the floor, 3 or 4 long, 8-inch poles along the passages, with split lagging or sawmill slabs across them. These support the roof when the ground caves, and enable the timber-sets of succeeding slices below to be placed without great difficulty.

In very treacherous vein-matter or under a bad roof, each cross-cut has risers at stated intervals that take the place of the level for removing the ore to the main level. The remainder of the work does not differ materially from that described.

Subdrifting and Caving System.—This system originated at the Brotherton mine in Lake Superior, and has been adopted with excellent results in many wide soft-ore mines. Cross-cuts are driven from the hanging- to the foot-wall, where they connect with longitudinal levels. At intervals of 50 feet risers *a* (Fig. 32) are driven 50 feet upward to the level above. The first six feet are cribbed and a couple of driven sets put on over them. Subdrifts *c* are then driven each way to meet the corresponding drifts from the adjacent risers. The riser meanwhile is continued another 6 feet upward, when timbers for a second subdrift *d* and above them for the third subdrift *e* are laid, until finally the riser reaches the level above at a distance of 50 feet. This divides the space between the two levels into three subdrifts, each having a top or back of ore above them 6 feet thick. The ore from subdrifts is delivered through a riser, and shipped to the cars on the lower level.

Following the completion of the subdrifts the capping of ore above the level *g* is next removed by dividing the ore into blocks of 10 feet square, beginning at the end of the level *g*. This weakens the pillars, which crush and permit the removal of the ore. That above the subdrift *e* is caved in a similar manner, followed by *d* and *c*.

Occasionally it is necessary to support temporarily the timbers above the ore, as at *h*, by long timbers, but in all cases all the close timbering is employed to control the pressure and prevent accident. In this method all timber can be lowered from the levels above and handled without interference with haulage. Ventilation is good, and simple means of escape are provided in case of accident.

Fig. 32.—Subdrifting and Caving.

Slicing and Filling System.—This system is practised in wide deposits, and consists in taking two slices of mineral in successive squares from a lower to a higher level and filling up the excavation with rock or other material quarried for the purpose. In this method the lifts are 60 feet each and the slices of attack 6 to 7 feet high, taken horizontally across the vein for as broad a face of attack as may be decided upon. Two main levels *c* and *d* (Fig. 33) extend to the end of the district to be thus mined, being at intervals connected by risers *e*. The first slice is removed at level *c*, and

waste rock from above delivered through the opening *e*, which is then spread to fill the excavation completely. The temporary level *f* is made on the top of the setting *g*. In each successive slice chutes are constructed for the purpose of letting down mineral into the lower level, from which it may be trammed to the shaft. This process is repeated until all of the mineral has been worked out in

FIG. 33.—A Slicing and Filling System.

slices represented by dotted lines and the waste rock distributed to fill the spaces.

In Fig. 34 is shown another method in which, if the exploration of the deposit is determined, this form of slicing-shaft, number 8, may be located permanently, and the rock-shaft, number 7, sunk. The drift *A* on the hanging-wall side of the deposit is connected by cross-cuts with the level *C*. Cross-cuts are also driven through the deposits for the drift *A* to the foot-wall and winzes sunk between the levels. The deposit having thus been blocked out, in each lift of 60 feet high there are three methods of removing the ore, according to the extent of the face, which may prove to be self-supporting.

First Method.—From the drift *A*, slices *DD* are taken 6 feet wide on either side of the timbering, cross-cut *B*, whose space is being filled with rock dumped into *A* through a winze from the

level above. The next two slices, *EE*, are taken while the excavations *DD* are being filled and the slices *FF* are mined, while *E* is being filled with rock. The filling material is lowered from the surface through rock-shaft number 7 and is hauled from the shaft to the winzes above.

This process is continued until a layer 6 feet thick has been removed over the entire deposit. The next slice above is operated

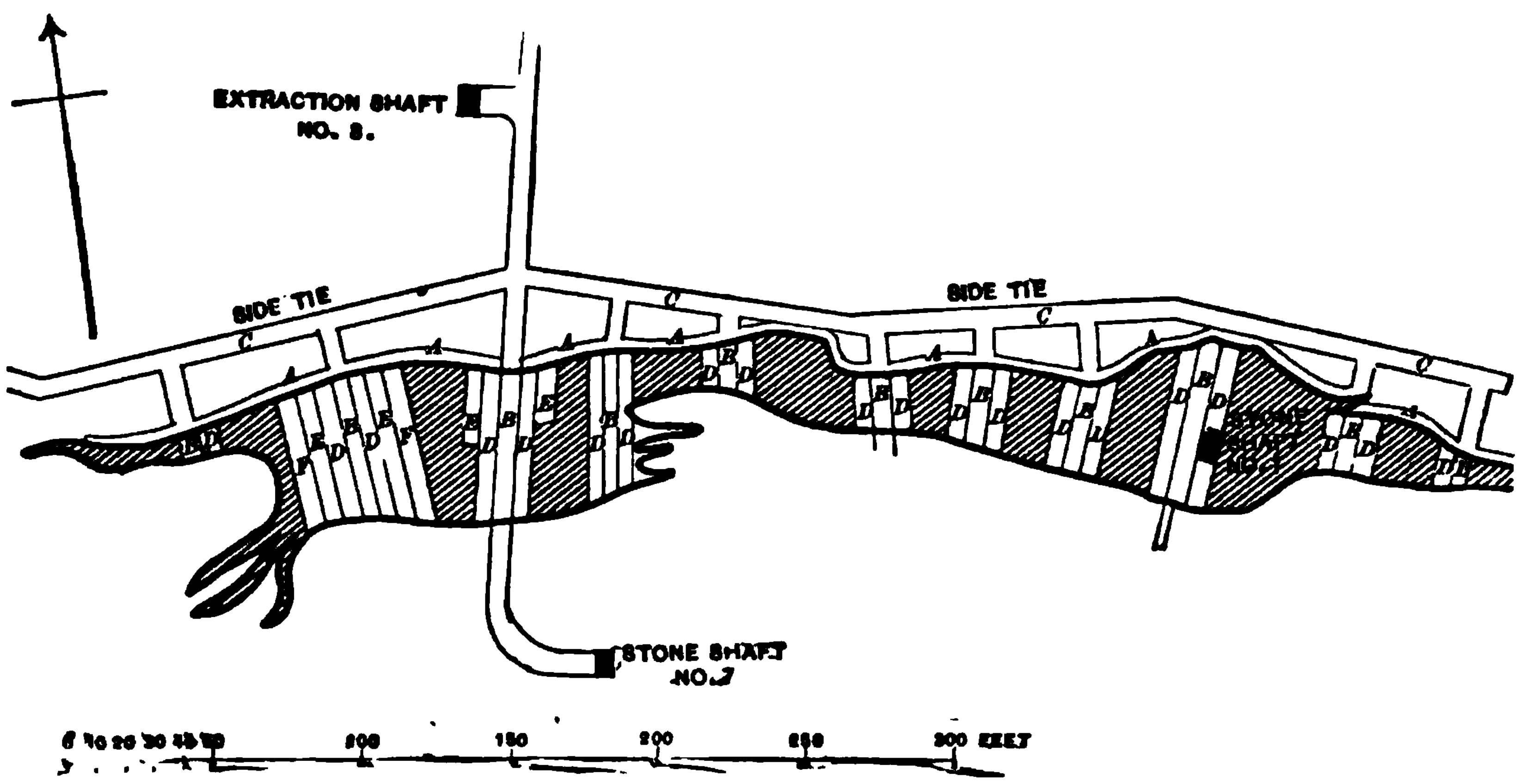


FIG 34.—The Filling System, Plan of One Level.

in the same manner, with the exception of a new drift driven above *A*, as in timbering as at *B*. The miners in removing the second slice stand on the waste while at work, and behind them is spread the material lowered for the purpose. The fillers follow the miners very closely. Mill-holes are timbered upward as mining progresses, the filling being packed about them.

Second Method.—When the vein-matter is dry and stands firmly it admits of a wider vein than 6 feet; the attack may be made from the foot of the wall side for a width of half the distance between the cross-cuts *BB*. The ore is then removed through the cross-cut levels while the filling is being spread as above described. The slices are driven in a direction parallel to the strike and across it, as in the first method. It may be well to mention that similar

operations are being conducted not only at other points on the same level, but also in a similar manner at other levels above or below.

Third Method.—This is designed for veins not over 20 feet wide. Here the face of the attack is the two sides of the pillars cut out by cross-cuts *BB*. The miners, starting from the cross-cuts, work toward one another, parallel with the strike, until they meet, their first cut being made at the walls, the filling progressing behind them as before.

This system of filling is considered a cheaper and safer system than either caving or square set for the extraction of mineral, whether soft or hard, in large deposits, particularly if a neighboring quarry can supply the waste rock. It is of the widest range of application. Very little timber is needed, and, with the exception of the first cross-cut *B* made in the solid ore, the mining is cheaply done, because the width of face is large and the miners have two free faces to take advantage of while blasting. Any material whatsoever will serve for filling, but the best is that mixed with clay, as it packs well. It should be in pieces as large as may be conveniently handled, for the "smalls" settle too much. In a few weeks' time the entire mass of filling compacts so that no difficulty is experienced when the stowing of the upper level is reached.

In the caving system, extreme caution is necessary to prevent sudden falls of roof, and the adoption of the method is only justified where there is an absence of filling material and a scarcity of timber. But if the vein-matter is comparatively firm, the method with filling presents greater security, and may be cheaper. In such event the choice of method depends upon the relative expense of procuring quarried rock and framed timber.

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CHAPTER IV.

POWER GENERATION.

The Power Plant.—The engine plant consists of boiler and engine, occasionally supplemented, in favored localities, by water-power electric plants. The plant must be installed as compactly as possible and of ample size. Common prudence suggests that the machinery be in slight excess of the immediate wants. This is more important, the lower the value of the mineral output. While the latest and best machinery will in the long run prove to be more economical, the question of finance and the amount of capital available is not to be ignored, and the engineer may be compelled to content himself with expedients until the mine has been fully developed.

The mechanical considerations will call for the following machinery: In a slope mine, stationary engines or locomotives for haulage ; in a shaft mine, an additional hoister; cutting machinery for coal, or drills for the rock or ore; fan for ventilation; pumps for the accumulating waters; machinery in tipple, breaker, or mill for crushing, screening, and jigging the mineral; pumps in the mill; electric lights for the mill and a portion of the mine. Either steam, electricity, or compressed air may be the motor agent in any of these cases; their comparative advantages and economy are discussed in Chapter VI.

The fan, the steam-locomotives, and such oil-engines as may be employed are alone independent of the central plant. Other machinery receives its power from the central source. Except the self-contained oil-engine and the rare hydraulic engine, all other motors depend for their primal power on the steam generator.

The Boiler Rating.—The boiler is essentially a device for converting water at low temperature into steam at a high temperature, the measure of its capacity being the quantity of water which it can evaporate in a given time. It must have sufficient surface exposed to fire to accomplish this end, hence the boiler is rated on its evaporative power and the area of its heating surface. The commercial requirements, demanding some definition of boiler unit commensurate with that of engines which it must supply, have accorded an evaporation capacity of 34.5 lbs. of water per hour from and to 212° as *equivalent* to a horse-power. The standard horse-power corresponds to 30 lbs. (3.55 gallons) of feed water evaporated from 100° F. to steam at 70 lbs. gauge pressure per square inch. This requires an absorption by the water through the boiler shell of 33,305 British thermal units per hour, practically equivalent to the centennial unit just mentioned. The general term of unit horse-power thus retained, though a deceptive one and unscientific, is convenient.

In calculating the size of boiler required for a certain purpose the amount of steam consumed per hour in the aggregate by the various engines may be known from table, page 118, from the given conditions of the operation. The amount of heat, measured in B.T.U., necessary to develop the given quantity of steam may be obtained from the steam tables. The number of boiler horse-power to give the required service is found by dividing the total B.T.U., necessary for evaporating the steam required per hour by 33,305 B.T.U.

This power may be subdivided in any manner if one boiler is added in excess of the average demand. It would be more advantageous to install the required heating surface for average load, remembering that any unit may easily be forced 33⅓ per cent without sacrificing efficiency. Then at times of emergency one unit may be repaired while the remaining units are overloaded, yet within their capacity. Thus, if 800 pounds of steam are required per hour as a maximum and but 3000 pounds throughout the day, three units may be installed. If 12,000 pounds of steam are in demand, four units may be installed, giving opportunity for one boiler to be

repaired while the remainder are forced. Thus the plant is not too large.

Practice has shown that to enable the absorption of 33,305 B.T.U. to be effected through the shell there must be a heating surface of 14 sq. ft. Hence 14 sq. ft. of surface exposed to the flames under a fire-tubular boiler constitute a boiler horse-power or 12 sq. ft. of exposed area in the water-tubular boiler.

The grate area must be sufficient to provide space for the coal and be at least $\frac{1}{50}$ of the total heating surface. In other words, each boiler horse-power must have from 0.24 to 0.28 sq. ft. of grate area. If the grate is too large, the temperature of combustion is too low. High temperature in the furnace is the first condition of economy, the second being that the heat produced shall be absorbed as completely as possible by plenty of clean heating surface. A grate surface too small causes clinkering of the coal. A low-grade fuel requires a large grate. As a limited quantity can be burned on a square foot of grate, the maximum evaporative capacity of the boiler is limited by its grate area.

The Standard of Boiler Comparison.—The equivalent evaporation of a boiler is that amount which would have been evaporated had the steam been produced at atmospheric pressure (=212°). It may be ascertained by dividing the B.T.U. necessary to boil the feed-water into steam by 966, the latent heat of steam at 14.7 lbs. pressure. Thus, feed-water at 160° F., converted into steam at 100 lbs. gauge pressure, requires 1057 B.T.U. The factor of evaporation is then $1057 \div 966 = 1.094$. The equivalent evaporation is 1.094 times the actual evaporation.

The rate of evaporation is about 8 to 19 lbs. of feed-water per pound of coal.

The Feed-water.—Care must be taken to secure ample supply of pure water for the boiler. Acids in the water corrode the boiler and reduce its efficiency and life. Salts in suspension or in solution give trouble in depositing within the boiler and forming a compact layer which is both injurious and wasteful of heat. A 100-horse-power boiler evaporating 3000 pounds

FIG. 35.—A Hoist-house with a Reel Hoister and Cornish Pump.

of moderately pure water per hour would in a year's time receive 248 lbs. of solid matter, which would cover the surface $\frac{1}{4}$ inch thick unless removed before being consolidated. Such a crust is

FIG. 36.—Water-tubular Boiler and Economizer.

inevitably formed in all boilers, and, being a poor conductor of heat, prevents heat from reaching the water. A scale of $\frac{1}{16}$ inch thick on the tubes causes a loss of 13 per cent of the fuel heat.

Moreover, the boiler-shell becomes excessively heated for the same steaming effects and tends to blister and warp. The scale may contract, and cracking, will suddenly expose the heated plate to the water; the generation of steam may follow and an explosion ensue.

Boiler Scale.—These two wasteful and dangerous sources require that care be exercised in the selection of the water-supply for the boiler plant. Mine waters are usually acid and should be neutralized by ammonia or precipitated by chloride of barium in the tank prior to being delivered to the boiler. Other sources of supply, except in the mountainous regions of the Rockies and Sierras, contain organic or mineral substances which should be rendered innocuous in the boiler or, preferably, removed before introduction by feeding the water-supply through the economizer or heater, Fig. 36, prior to its delivery to the boiler. A considerable deposition will ensue from mere increase in the temperature of the water; the sediment there formed can readily be removed. Such material as still remains in suspension in boiler must be rendered innocuous by an addition of kerosene, which would cover the grains with a film and prevent the sediment compacting too densely on the tubes and shell. Washing-soda will dissolve the scale and change its nature. The amount of kerosene to be fed should not exceed 2 quarts per week per 100 horse-power, and of soda one pound per week. An excessive use of these remedies would cause "foaming" and deliver wet steam into the cylinder. The best remedy for the formation of scale is to blow off and to clean the boiler regularly every week.

When the condenser is used in a steam plant its supply is returned to the boiler, and if lubricating-oil has been used excessively in the engine a very injurious mixture is introduced into the boiler. The oil should be absorbed from the steam before it reaches the condenser, or removed from the water leaving the condenser before delivery to the boiler. It is caught on screens or separated by a centrifugal machine.

It is a question of no small matter for the engineer whose water-supply is chemically bad to determine whether to incur

the expense of a special distillation plant, or to employ the condensed exhaust-steam. Some engineers have solved the problem by using the very smallest amount of lubricating-oil possible in the pumps and engines, thus saving the boiler at the expense of wear upon the pistons. The author favors the plan of purifying by chemical or thermal process such water as requires it in a feed-water heater, or an economizer, before delivery to the boiler. It may even be desirable to sink an Artesian well for water-supply if not otherwise attainable, though in that event it should not be drawn from a limestone formation.

The Feed-water Heater.—The feed-water heater is designed to utilize the heat in the exhaust-steam or the flue-gases by bringing the feed-water supply in close contact with them. It is known as an economizer, because only waste heat is employed for the purpose. The simplest form of heater has a tank of matched dovetailed boards in which is placed the coil of pipe, one end open, blowing into the atmosphere, and the other end connected with the exhaust from the engine. The heat in the steam is communicated to water and its temperature raised, while at the same time the carbonic acid or carbonates in the water are released and the salts deposited. The sediment can readily be removed, if a hand-plate be provided near the bottom.

A duplicate tank to this may then receive the exhaust-steam during the process of cleaning. Hay is often placed in the water-supply tanks, filtration through which would remove some of the muddiness.

Feed-water heaters can also be procured of the closed type, consisting of a steel cylinder filled with tubes through which the steam flows, while the water circulates around them or the reverse. The benefit derived from the use of the economizers increases with the wastefulness of the boiler, and the need of purification warrants their introduction when the water-supply is chemically bad. The life of a well-built economizer is from fifteen to twenty years. The average plant will save at least 10 per cent of the fuel value. The gross return on the investment of the economizer is 48 per cent annually with coal at $5

per ton, and 19 per cent with coal at $2 per ton, assuming twenty hours of service out of the twenty-four. In addition to the saving of heat and the perfecting of the waters, there is another advantage, that the boiler receives water at a high temperature and is saved from the straining effects which result from cold water. A small pump, or an injector, delivers the supply to the boiler. The latter is cheaper, equally reliable, and easily handled.

Types of Boilers.—It is a matter of considerable difficulty to select the best boiler, because of the great variety now on the market. The three types have each their advocates. The Cornish type of boiler, with the fireplace in two large tubes, has a high efficiency, and is much liked where its length is no objection. The multitubular boiler, with the fireplace below the front end and fitted with numerous tubes through which the fire returns on its way to the chimney, is a most common type. The latest type is the water-tubular boiler (Fig. 36), in which the fire and the gases circulate around the tubes; in these the water flows and is converted to steam. These three types have their separate fields of usefulness and find wide acceptance in engineering circles. Their records can easily be had, so they must be selected according to local requirements. The relative efficiency of the three types is 1, 1.6, and 2; their relative initial cost is very nearly in the same proportion.

The Cornish boiler utilizes 50 per cent of the fuel value in the production of steam; the fire-tubular boiler absorbs 60 to 65 per cent of the heat generated in the fireplace; and the water-tubular boiler absorbs fully 80 per cent of the fuel value. The comparative durability of the three types is about the same, with an advantage in favor of the more expensive water-tubular boilers. They all are good steamers, though the last-named can be most readily forced when the emergency arises.

Water-tubular Boilers.—The steam pressure to which these types may be worked is respectively 60 lbs., 100 lbs., and 200 lbs. pressure per square inch, as a maximum for ordinary conditions. Inasmuch as the simplest method of increasing the power of a steam-engine consists in raising the pressure of the boiler

steam, it follows that when such emergency arises the water-tubular boiler will best fit the case. These boilers are sectional and can be easily transported. They are safer than the ordinary type even at higher steam pressure, because the excessive pressure beyond the danger line will overstrain but one tube instead of the entire boiler of the other types. They are quickly cleaned, easily repaired, and give drier steam and with a smaller amount of fuel than the other types.

Heat Losses.—All the heat developed by coal must be absorbed in steam production. There are, however, three sources of loss—that due to heat carried by the flue-gases escaping up the chimney, the radiation from the walls and shell of the boiler, and the heat remaining in the ashes.

The waste in the ashes depends on the conditions in firing, etc. Their weight may be 20 per cent of the total fuel burned, and in them is also some unburned combustible. The unprotected boiler surface radiates, per square foot, 650 B.T.U. per hour. Thus the loss from a 100-horse-power boiler and its wall is equivalent to the heat of 100 tons of coal a year, or at least 15 per cent of the fuel. A covering of magnesia, hair-felt, or asbestos, 1 inch thick, would in five years' time save much more than its cost, which is 25 cents per square foot. A brick arch over the boiler, furnishing a conduit for flue-gases on their way to the chimney, would be still better. Boilers in batteries, if placed back to back instead of facing one another, would also reduce the radiation loss, though the cost of stoking would be increased.

Flue-gases.—The flue-gases have a temperature between 300° and 600° F., being higher with the forced draft than with the chimney draft. In order to give place for the air-supply in the combustion of the coal, these flue-gases are removed as promptly as possible. Each pound of coal produces gases that transport 1500 or more B.T.U. This is always more than 10 per cent of the total available heat in the fuel. The only recoverable portion of it is that which the economizer may utilize. Smoke increases the heat loss by that which is latent in the unconsumed combustible.

The Air-supply.—The aggregate of the above sources of heat loss is at least 15 per cent of the fuel value, and is more frequently 35 per cent in the ordinary boilers.

Economic combustion will reduce the great waste of heat by providing the proper supply of air, the perfect burning of a well-selected fuel, and a good regulation of the draft. Conditions for complete combustion require that the particles be brought into intimate contact with the requisite amount of oxygen. The amount of air-supply being the chief consideration, each pound of coal will require an amount of air at least equal to $A = 1.52(C+3H-0.4O)$, in which C, H, and O are the percentages of carbon, oxygen, and hydrogen.

This is about 160 cu. ft. or 12 lbs. of air. More than this is necessary to insure intimate contact of the fuel constituents and the air. Hence 24 lbs. are usually given for each pound of fuel with the chimney draft and about 18 lbs. with forced draft. Any excess over this amount chills the fire and will only carry heat from the furnace to the chimney. With less air the carbon will develop only 4450 B.T.U. per pound, instead of 14,500 B.T.U., which is more wasteful than smoke. The production of CO instead of CO_2 results from a deficiency of air; the fire is dull and sluggish.

An occasional analysis of the flue-gases, by the D'Orsat apparatus, for free oxygen and CO, will reveal the working of the furnace. The former indicates air excess, the latter air deficiency.

Calorific Value of Fuels.—The heating power of a fuel depends upon its combustible elements and the amount of ash. The fuel should be bought on its calorific value and according to analysis. This latter may be the ultimate analysis, giving the percentages of carbon, oxygen, and hydrogen in the fuel, whence its calorific value is $P = 14{,}550C + 62{,}000H - 5400(O+N)$; or an approximate analysis giving the amount of volatile matter and fixed carbon in the fuel. In the latter case, assume M to be the percentage of volatile matter, V, compared with the dry combustible, $M = \frac{100\,V}{V+C}$,

and A a coefficient whose values are given below. Whence the calorific power: $P = 146.7C + AM$.

$A = 252$ for $M = 2$ to 12;	$A = 169.2$ for $M = 30$ to 35;
$A = 216$ for $M = 12$ to 17;	$A = 144.0$ for $M = 35$ to 38;
$A = 198$ for $M = 17$ to 24;	$A = 142.2$ for $M = 38$ to 40;
$A = 183$ for $M = 24$ to 30;	$A = 136.8$ for $M = 40$ to 50.

The heat which can be developed from a pound of coal varies between 11,000 B.T.U. and 15,000 B.T.U. As one pound of water requires for its evaporation and conversion to steam something over 1100 B.T.U. it follows that from 10 to 14 lbs. of steam can be produced for each pound of coal, though some boilers realize only 4 to 6 lbs.

Undoubtedly the semi-bituminous coals are better steam-producers than anthracite coals, the amount of the ash being less. But it is usually a question of local price as to which is preferable in a given case. A superior coal is that which has about 85 per cent of fixed carbon with some volatile matter to assist in the firing. A steaming coal should kindle readily and burn steadily without clinkers.

Slack is nearly equal to coal in its calorific value, but has usually too much refuse to be acceptable, except where the transportation is cheaper. The size of the fuel is selected according to the relative grate openings. The smaller coals and slack are cheaper than the larger size of the same coals, but the amount of ash in the former increases the labor of handling.

The Rate of Combustion.—The amount of coal that can be burned depends upon the intensity of the draft and the air-supply. The thickness of the bed should be as small as possible without requiring excessive watchfulness. A 5-inch bed of anthracite pea coal or an 8-inch bed of lump coal is good practice. Only 12 lbs. of anthracite or 15 lbs. of bituminous coal can be burned per hour per square foot of grate area by natural draft with economical results. The results from the boilers depend more upon the character of firing than on any other single element.

The Methods of Hand Firing usually adopted in practice with ordinary grates are the "spreading" method, in which a

thin layer of coal is worked over the entire area from the bridge towards the front; the "alternating" method, in which the fresh coal is charged alternately on either side of the fire-box; and the "coking" system, for bituminous coal only, that charges it on the dead-plate at the front of the fire and pushes it back, when coked, into the grate to make room for a new charge. This is advantageous with all long-flame gaseous coals. Two stokers and two engineers can fire 35 tons of coal per week under the boilers during 24 hours. The limit for one man firing boiler and running the engine is about 10 tons of coal per week for a 50-H.P. engine.

Stokers.—Mechanical systems of feeding hard coal are employed to advantage. The Steam Users' Association of Boston reports that in all plants stokers may prevent smoke, save coal, and reduce labor—as much as 30 per cent in big plants employing also coal-handling machinery. Mechanical stoking reduces the smoke nuisance by furnishing a continuous charging. Stokers, however, cut down the capacity slightly, though they respond to a sudden demand for steam as well as hand firing. The amount of repairs is not excessive.

The stokers are travelling grates or moving bars fed from a hopper continuously at the front and carried by power toward the rear at a slow rate such as to insure a complete burning of the fuel before it reaches the rear of the fire-grate where it is dumped. Fine sizes of coal can be utilized and little power is required. Undoubtedly they afford a betterment of evaporative results, but it is a question whether the same amount of improvements cannot be effected in other directions.

Fuel Consumption.—The consumption of fuel per hour averages $3\frac{1}{2}$ lbs. per horse-power with a common boiler and ordinary engine. The very best results which have yet been obtained in practice require 1.5 lbs. of good coal to produce a horse-power hour.

The question of fuel economy is an exceedingly important one, whether one considers the production of power at the coal-mine, where the fuel is cheaper, or at the metal-mine, where it is very dear. Any economy effected in this direction represents a

corresponding gain. The bituminous collieries of the United States consume between 1.5 per cent and 2 per cent of their total output in providing power for pumping and water. The anthracite coal-mines of the United States are estimated as consuming from 8 per cent to 10 per cent of their total production for the same purpose.

A cord of well-dried spruce is capable of driving a common slide-valve hoister about 320 hourly horse-powers; a ton of lignite, 470; a barrel of petroleum, 170; a ton of anthracite, 650.

Weathering Coals.—Fuels should be properly housed from the weather, and not piled out of doors, to prevent absorption of moisture, the evaporation of which causes deterioration. Freshly mined coal absorbs three times its volume of gas or air, and this absorption produces a slight decomposition that reduces its heating power. Moreover, spontaneous combustion may develop from a prolonged oxidation, which can be prevented only by a thorough ventilation of the coal and its protection from the elements. The practice of sprinkling the coal arrests the action temporarily, but deterioration soon begins, which would be facilitated by the presence of fine coal. It is a good practice to have a storage for 500 lbs. of coal for each indicated horse-power of engine plant.

Liquid Fuels.—Petroleum has an approximate analysis of carbon 85 per cent, hydrogen 13 per cent, and oxygen 1 per cent, and a calorific value of about 20,000 heat-units per pound. This fuel can be injected into the fire-box from tanks by a nozzle with the aid of compressed air. This blast induces a current, atomizes the oil, and delivers a spray with an excess of oxygen which permits the production of an intense flame. No grate is required. One pound of oil equals 2.18 lbs. of coal in calorific value. Where conditions favor it, gas from coke-ovens can be employed to advantage under a boiler.

Draft.—A draft is necessary to carry off the products of combustion and to admit a fresh supply of air to the fire. The force of the draft is measured by a water-gauge which varies between 1 inch and 2 inches. It may be either natural or forced.

Chimneys.—Natural draft is procured by a chimney of brick or of sheet-steel tubing. It depends upon the relative densities of the hot gases inside the chimney and the external air. The difference in the temperatures and the height of the stack will determine the velocity of the flow and the pressure of the draft. The inside area determines the volume of air which can be carried. This is known from the total hourly consumption of fuel, F, which is based on 5 lbs. per horse-power. The desired horse-power being known, the effective area, E, may be determined for a known height of chimney, H.

$$A = \frac{0.0625F}{\sqrt{H}}.$$

The effective area E is less than the actual area by the amount indicated, $E = A - 0.6\sqrt{A}$.

The commercial horse-power of chimney is $3.33E\sqrt{H}$.

The first cost of a brick chimney is greater than that of a sheet-steel stack, but its stability is greater and its repairs are less. By the use of a damper regulated by hand or automatically by steam, any lower pressure than the maximum draft for which it is designed may be maintained in the latter arrangement. A small diaphragm raises the lever and closes the damper when the steam pressure in the boiler exceeds a certain desired limit.

Blowers.—Forced draft may be obtained by the use of a blower driven by the engine and dynamo or belt. By it air is delivered through the pipes into the fireplace at a pressure of perhaps 10 lbs. above the atmosphere. The steam-jet may also be employed, acting like an injector; though it is not an economical device, it softens the clinkers and aids combustion. A steam-blower can be built composed of a horizontal ring with holes punctured in it. Steam from the boiler is discharged through the openings and creates a current that will increase the draft and thereby the capacity of the boiler 20 per cent to 30 per cent. The ring is made of 1¼-inch pipe 12 inches in diameter punctured with eighteen ⅛-inch holes for a 60-horse-power boiler. Such a blower costs 90 cents an hour for operation.

Exhaust-fans may be so placed as to draw the gases out of the furnace and deliver them into short stack to aid draft.

The chimney has a higher first cost than any of these systems which require short stack only, but is cheaper to maintain. It is, however, very wasteful of heat, since the flue-gases must be hot in order to produce draft. Nevertheless, the latter adapts the boiler to a wide range of power, for the fire can be made more intense.

Boiler Installation.—The boilers should be on a level below the engine-room and set as high up as is consistent with the fireman's duties. The coal should be stored above them and gravitation availed of for feeding the boilers and for removing ashes. This would be an ideal condition as to labor. The cost of power production is not always a matter only of fuel selection and economy of combustion, for the cost of handling is inseparably connected with it. Generally speaking, the annual coal consumption equals the initial cost of the boiler plant, and economy in labor may easily be effected equal to 10 per cent of the initial cost.

With the pipes hung to drain towards the boiler, covered with a layer of asbestos, magnesia, or felt covering, with a steam-trap properly placed, a supply of dry steam will be delivered to the engine. Each square foot of exposed surface of pipe radiates 1.8 B.T.U. per hour for each degree Fahrenheit difference in temperature between the steam and the external air. Thus a 4-inch pipe carrying steam of 90 lbs. pressure (320° F.) 40 feet long radiates an amount of heat equivalent to 17 lbs. steam from 120° F. to 90 lbs. absolute pressure. Hence a considerable loss of power ensues if pipes be uncovered Every 284 feet of 4-inch pipe covered with 1-inch layer saves a horse-power.

As the allowable velocity of steam in pipes is ordinarily 50 feet per second, it is necessary to have a line of pipes ample in area to furnish a copious supply of steam with but small loss. A 4-inch pipe will discharge 115 lbs. of steam per minute, the initial pressure being 100 lbs. gauge, or it will deliver 109 lbs. of steam at

80 lbs. gauge. A 2-inch pipe will deliver 30 lbs. and 24 lbs. of steam respectively. As it requires about 25 lbs. of steam to supply one horse-power per hour, a 2-inch pipe will serve for about 60 horse-powers.

The maximum absolute steam pressures for successful operation are ordinarily as follows: For simple non-condensing engines, 60 lbs. gauge pressure; for compound non-condensing, 100 lbs. gauge; for simple condensing, 90 lbs. gauge; and compound condensing, 130 lbs. gauge. The character of the boiler will largely determine the maximum pressure available. A multitubular boiler cannot be forced beyond 100 lbs. pressure, while the water-tubular boiler may be carried to 200 lbs. or more.

The Steam-engines.—The steam will be conducted to the underground pumps, the exhaust-fan or the ventilating steam-jet, and to the steam end of all compressors, electric generators, and hoisters. Its behavior in the cylinders of these engines is alike in all, and the following considerations will apply to all.

Classification of Engines.—The engines may be classified according to the manner in which they utilize the elastic force of the steam. Direct-acting pumps and other engines having a constant resistance do not admit of a change of pressure within the cylinder of the engine; but where the resistance varies, or it is possible to provide for the varying resistance, the steam is permitted to expand, with the corresponding reduction of pressure, with marked economy.

The Behavior of Steam.—Fig. 37 illustrates the behavior of steam inside of a pump in which the length of the diagram corresponds to the length of the cylinder, and the height of the diagram the pressure of steam at various points. The upper horizontal line represents the steam pressure admitted to the cylinder, which, remaining constant, will be straight. The atmospheric pressure being also constant, the former will be parallel to it and at a distance above it according to the scale on which the pressure may be drawn. The pressure of the atmosphere being 14.7 lbs. per square inch, a line may be drawn below it at that

scale and represents the line of absolute zero of pressures. *OX* is that line, *MN* the atmosphere line, and *AB* the line of constant steam pressure. When the piston reaches the end of its stroke the exhaust-valve is opened for the discharge of the steam. The pressure falls nearly to atmospheric pressure. On the return stroke of the piston the steam is being expelled at a constant pressure, represented by the line *CD*, until the piston is at the end of the cylinder. Steam is now admitted. The pressure rises and the line *DA* represents such increase in the pressure.

On the other side of the piston, the operations ensue in the same order, the line *CD*, representing the back pressure, on one side being simultaneous with the steam pressure, *AB*, on the other

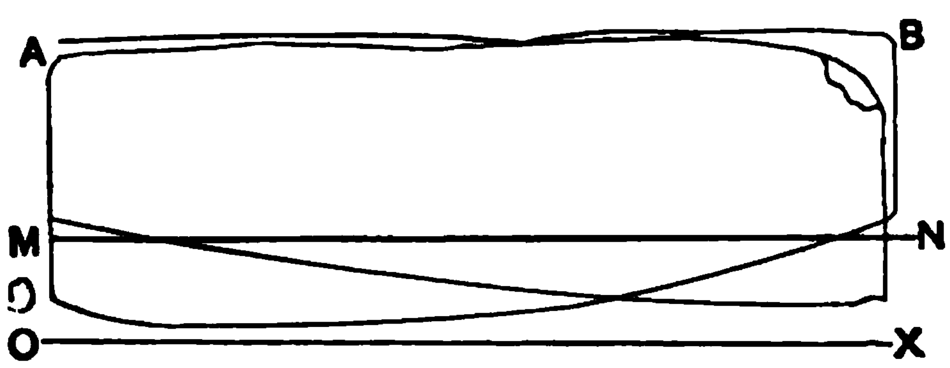

FIG. 37.—Indicator Card from a Non-expansion Steam-engine.

side. If the steam be exhausted into the condenser, the line *CD*, representing the back pressure upon the pump, will be below the atmospheric line *MN*. Its length above the zero line, *OX*, is an amount representing the vacuum pressure in the condenser. This line would then be at about 5 lbs. as a minimum.

If the steam after being admitted at constant pressure for a portion of the stroke is cut off, the piston will continue to move in its stroke, but with a continual reducing pressure to the end. Fig. 38 will represent in the two lines *AB* and *BI* the behavior of the steam during admission and expansion. The back-pressure line, *CD*, will remain the same as in the pump. The discharge of the steam may be cut off at some point, *D*, on the return stroke; the steam still remaining in the cylinder will be compressed and the line *DE* will indicate the increase in pressure for the balance of the stroke. When admission occurs the line *EA* will rise more or less promptly, according to the freedom of entry of steam. The back-pressure line will be below or above *MN* according

to whether a condenser is attached or not. The diagrams represent, therefore, the behavior of the steam at all periods of the piston movements in the cylinder, and the area enclosed between *ABICDEA* measures the work done during one stroke. The vertical lines represent the pressures to scale, in pounds per square inch of area piston; the horizontal line represents the length of stroke, *s*. The area divided by the length of card and multiplied by the scale of vertical pressures equals the mean effective pressure, M.E.P., in the piston during the entire stroke. Diagrams such as these are called indicator cards and measure the indicated horse-power of the engine. The device known

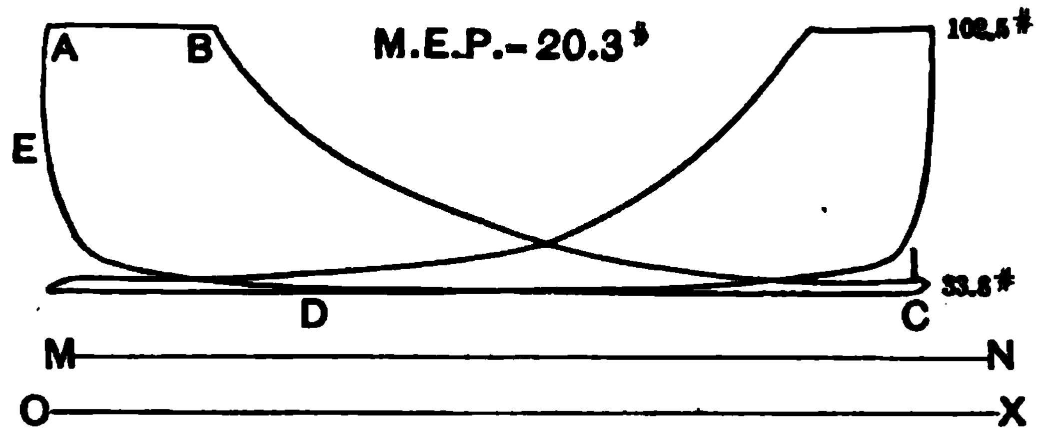

FIG. 38.—Indicator Diagram from a Non-condensing Engine or the High-pressure Cylinder of a Compound Engine.

as the indicator (Fig. 40) can be procured, by which such cards of steam behavior can be automatically registered by the engine while at work.

In order to obtain the most economical consumption of steam, the lowest point, *I*, of the expansion line, *BI*, should be as near to the back-pressure line as possible. The limit, however, is 3 lbs. gauge in non-condensing engines and about 5 lbs. absolute when condensers receive the exhaust.

The expansion line, *BI*, follows practically the equation of the equilateral hyperbola, $PV = \text{constant}$; if the cylinder walls be perfect non-conductors, the expansion line would be represented by the adiabatic equation $PV^{1.135}$.

The most desirable point of cut-off then is fixed, the conditions being given. Thus, if the pressure at *I* be 18 lbs. absolute and the admission pressure be 90 lbs. absolute, the cut-off will be at one-fifth of the stroke. The ratio of expansion is then 5; that

is, the volume of steam delivered to the cylinder up to point of cut-off is expanded to five times this volume at the end of the cylinder.

The Mean Effective Pressure.—By the following equation M.E.P. may be ascertained for a given admission gauge pressure, P, and a back pressure, B, in pounds absolute per square inch. By gauge pressure is understood the pressure above the atmosphere as recorded on the steam-gauge. Absolute pressure is the pressure above absolute vacuum. Atmospheric pressure is 14.7 lbs. absolute. $p=C(P+14.7)-B$.

The value for C is obtained from the following table for given ratios of cut-off.

VALUE FOR THE CONSTANT, WITH VARIOUS CUT-OFFS, FOR DETERMINING THE M.E.P.

Percentage Cut-off Apparent.	Ratio of Expansion, No Clearance.	C.	Ratio of Expansion with 3 Per Cent Clearance.	C.	Ratio of Expansion with 7 Per Cent Clearance.	C.
0.1	10	.3302	7.69	.4113	5.88	.4715
$\frac{1}{8}$	8	.3850	6.45	.4435	5.128	.5137
$\frac{1}{6}$	6	.4653	5.102	.5155	4.23	.5774
$\frac{1}{5}$	5	.5219	4.3478	.5682	3.7037	.6225
$\frac{1}{4}$	4	.5966	3.5714	.6368	3.125	.6861
$\frac{1}{3}$	3	.6995	2.754	.7294	2.481	.7708
0.4	2.5	.7664	2.3255	.7931	2.1276	.8242
$\frac{1}{2}$	2	.8465	1.8867	.8658	1.754	.8925
$\frac{5}{8}$	1.6	.9180	1.527	.9310	1.439	.9481
$\frac{3}{4}$	1.333	.9656	1.282	.9743	1.2195	.9842
$\frac{4}{5}$	1.25	.9785	1.2875	.9807	1.149	.9914
$\frac{7}{8}$	1.143	.9918	1.104	.9941	1.058	.9961

For all practical purposes the ratio of expansion and the point of cut-off are reciprocals of one another. The clearance space is left at the end of the cylinder, which the piston never fills and in which steam is allowed to accumulate and act as a cushion to prevent the piston from striking the end of the cylinder. This clearance space contains steam which is ineffective and changes somewhat the ratio of expansion for a given cut-off. The clearance is from 3 per cent of the stroke in the better classes of engines to 7 per cent in the simple slide-valve engines. High-speed engines may have a still larger clearance.

The shorter the point of cut-off the greater the efficiency of

the engine. The limit, however, depends upon the character of the admission-valve. With a simple slide-valve engine the cut-off cannot be less than 0.4 of the stroke. With the Corliss valves, or some forms of rider cut-off valves, the cut-off may be very much shorter, though it rarely is less than one-fifth in a single cylinder.

The M.E.P. and the area of the card in Fig. 37 may be greater than in Fig. 38, but it must be remembered that the volume of steam used in the former case is equal to the entire volume of the cylinder, and in the latter case to the volume up to the point of cut-off. Hence the work performed per pound of steam in the latter case will be greater than in the former.

The Use of the Indicators.—An indicator, which is essentially a stethoscope to the engine, reveals the character of the steam expansion compared with the theoretical; the amount of water existing in the cylinder at any moment; the point of cut-off; the setting of the valves; and the amount of resistance to exhaust. None of these can be detected from the behavior of the engine. Leaks, tight stuffing-boxes, loose piston packing, or a priming of the admission steam are likewise revealed by the indicator. The machine is simple, easily attached, and requires no elaborate calculation, and by the aid of the planimeter (Fig. 39) it is

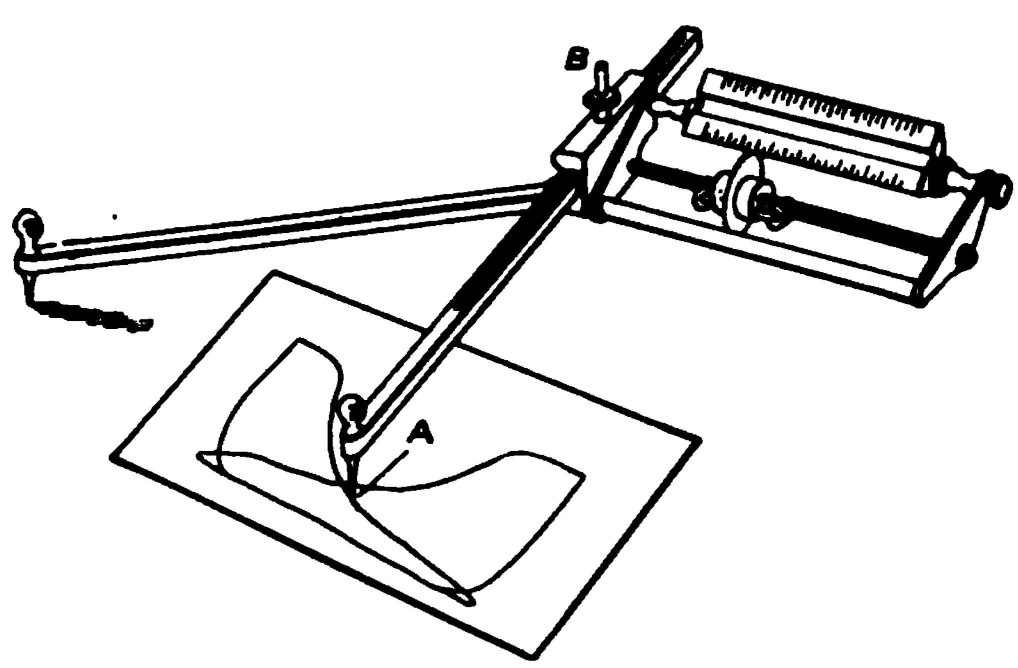

FIG. 39.—Measuring the M.E.P. of an Indicator Card by Planimeter.

possible to obtain a card by which to determine the indicated horse-power. Fig. 40 illustrates one attached to the engine.

An indicator is a device for registering the behavior of the steam within the engine cylinder. It consists of a drum and a

cylinder; the latter has a plunger whose rod carries a pencil above it and is connected by a pipe to the engine cylinder. When desired, the steam may be admitted from the engine under the plunger, which is then lifted an amount depending upon the pressure existing in the engine at that instant. With it the pencil rises, and each change of steam pressure will be indicated by a corresponding rise or fall of the pencil. To control the amount of lift of the piston and its pencil a spring of known strength is inserted above the plunger, and thus not only each change of pressure but also the amount of such pressure may be recorded. Each rise of pressure lifts the plunger a definite amount by compressing the spring correspondingly. A drop in the pressure is indicated by an extension of the spring. The pencil then rises a certain amount when the steam pressure in the engine and on the plunger is 100 lbs. per square inch. It will rise half that for a steam pressure of 50 lbs., etc.

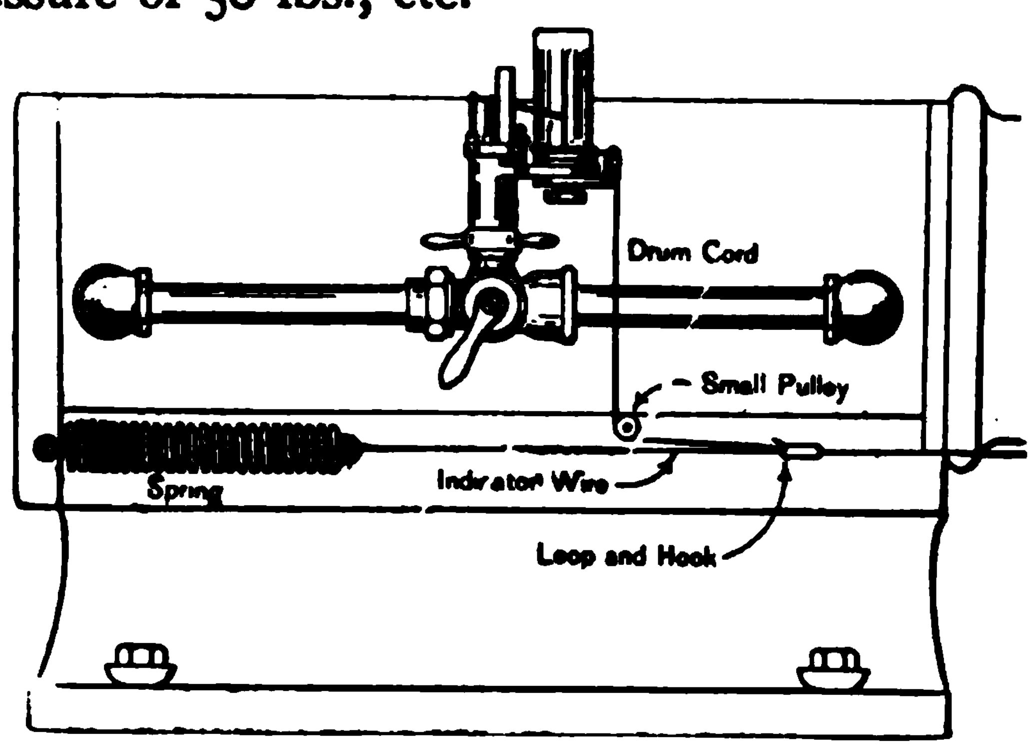

FIG. 40.—Attaching an Indicator.

The pencil bears against and records upon a sheet of paper, about 6 inches long, which is slipped on a drum of about 2 inches in diameter. A large wheel attached to the drum, turned by a string fastened to the cross-head of the engine, pulls the string, turns the drum, and the paper moves under the pencil. As the piston travels in its cylinder the pencil rises and falls simultaneously with it, thus furnishing a diagram called an indicator card. Various devices are employed to reduce the long piston stroke to a card length of about 4 inches.

The plunger springs are selected according to the boiler pressure, so that the height of lift of pencil shall not exceed 2 inches; thus for 100 lbs. admission pressure a number 50 spring is taken, which means that steam of 50 lbs. pressure will compress that spring an amount that will lift the pencil 1 inch; each change of 1 lb. steam pressure will move the pencil $\frac{1}{50}$ of an inch. The scale of the spring is 50. On the diagram drawn by such a spring the vertical lines represent the pressure at the rate of 50 lbs. per inch of height, and the horizontal line the stroke of the piston on a scale equal to the ratio of the piston stroke divided by the card length.

Steam Condensation in the Cylinders—Remedies.—The objection to a short cut-off and a high rate of expansion lies in the fact that the walls of the cylinder are not non-conductors, which, being comparatively cool at the end of the stroke, condense some of the entering steam whose energy is thus lost. The greater the ratio of expansion the larger is the percentage of the steam condensed. It may reach as much as 50 per cent during some portion of the stroke in some engines, and is always 25 per cent with unprotected cylinders. This is reduced by wooden lagging around the cylinders or steam circulating in an annular space around the cylinder. Another remedy consists in superheating the steam after leaving the boiler by passing it through the pipes heated by flue-gases before delivering to the engine.

A better remedy is to divide the expansion into several stages by compounding the cylinders. In this the steam, after partial expansion in one cylinder, is delivered to a second, and there expanded to the lower limit determined by the pressure in the condenser or the atmosphere (Fig. 41). The action in the combined cylinders is the same as in a single cylinder with the same ratio of expansion.

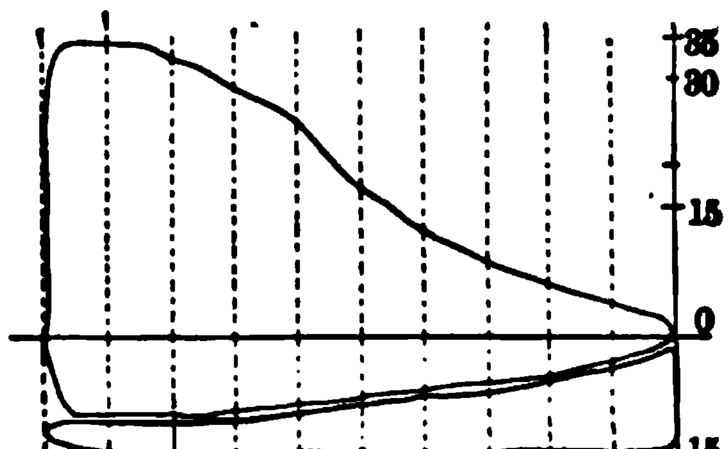

FIG. 41.—Combined Indicator Cards.

The Compound Engine.—These cylinders may be connected

on the same piston-rod and have a common stroke (tandem), or are side by side on different rods (cross-compound) and move in opposite directions. If, however, the motion of the pistons is to be at some other phase, a receiver is inserted between the two cylinders, the exhaust from the high-pressure accumulating in it before being discharged into the low-pressure cylinder.

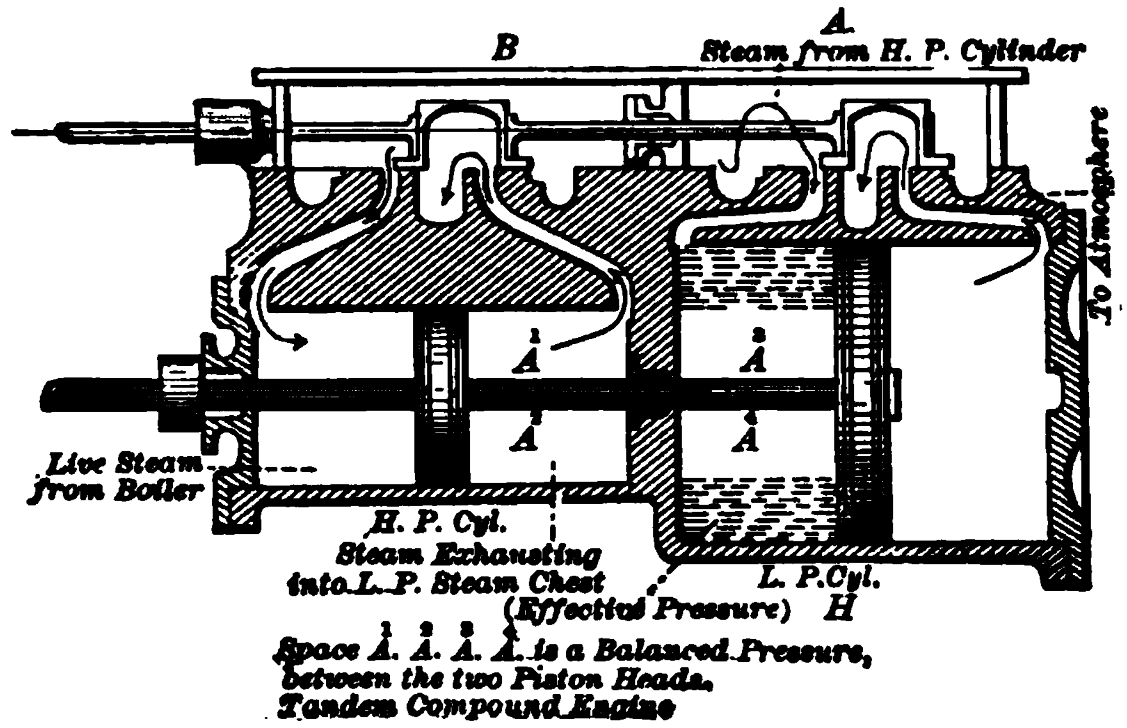

FIG. 42.—The Section of a Compound Cylinder.

The compound engine economizes steam by permitting a larger ratio of expansion, having smaller clearance volumes and far less condensation in the cylinders. The cost of the compound is, however, higher than the simple, its bulk and weight are greater, but its steam consumption is very much less.

Governors.—To control the engine and to economize steam by supplying to the cylinder only the amount that is necessary to do the work, two forms of governors are employed. The steam pressure may be reduced before admission to the cylinder by closing the valve in the steam-pipe; or an equally automatic regulation may be had by reducing the volume of the steam delivered to the cylinder by hastening the point of cut-off.

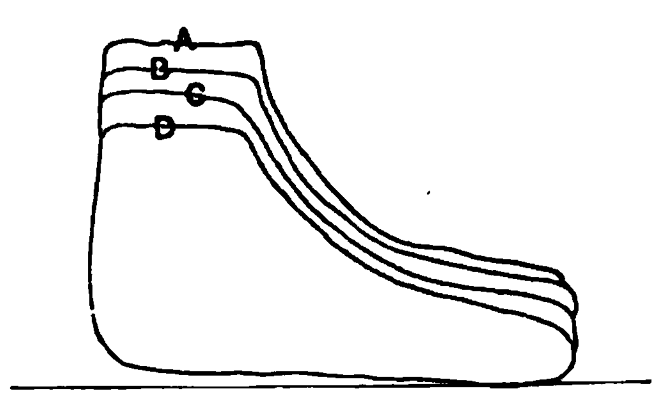

FIG. 43.—Indicator Diagrams from a Throttling Steam-engine.

In Fig. 43 are the diagrams obtained successively in the throttling-engine as the steam-supply is reduced. Fig. 44

shows the diagrams from a cut-off engine. The throttling is obtained by a pendulum governor connected with the driving-shaft of the engine, which closes the valve when the rate of revolution exceeds a certain limit. The cut-off is operated by some form of gearing connected with the shaft, and when the rate is too high reduces the travel of the valve. The throttling-engine is cheaper and suffers less from cylinder condensation than a cut-off engine, but does not receive so sensitive or so uniform a regulation. All engines have an independent throttling control for the engineer.

Automatic Cut-off.—The mechanism for shutting off the steam-supply before the piston reaches the end of its stroke may be fixed in position or variable, automatic, or adjustable. The fixed cut-off is intended for constant duty, the governor being set to shut off the steam-supply when the desired point is reached. The variable cut-off is needed for a varying load during successive strokes. In the high-speed engine the shaft-governor varies the length of travel of the slide-valve, and thereby hastens cut-off when the load is light. In the Corliss engine the governor operates directly upon an arm which varies the time of cut-off by changing the tripping position. The latter two types are automatic in their action, the Meyer being adjustable. Its rider-valves (Fig. 44) can be set at each end at any time to the desired cut-off without affecting the movement of the lower valve which determines the duration of the exhaust. The Corliss valves are also adjustable by changing the length of the rods from the wrist-plate to valve-spindles.

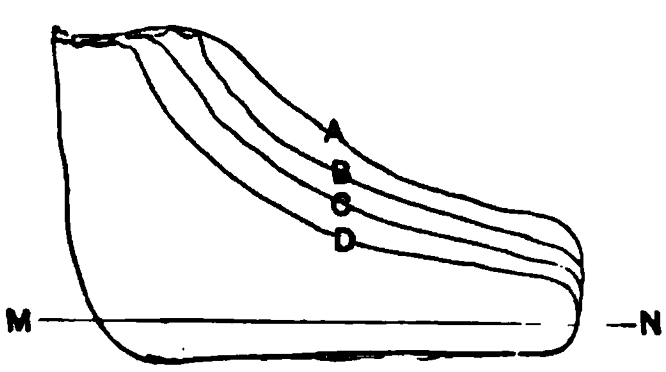

FIG. 44.—Indicator Diagrams from an Automatic Cut-off Engine.

Throttling governors are usually employed on all low-speed engines and shaft-governors on high-speed engines. Engines arranged to run "over" are smoother than those turning "under." Facing the engine broadside with cylinder on the left and crank-shaft at the right, right-handed revolution is called "over."

The Sliding Steam-valves.—The valves for regulating the admission and exhaust of the steam are of different types. Simple engines have one slide-valve to perform all the functions for both ends. Such a valve admits of a cut-off not shorter than 0.4, and wastes considerable power in supplying the steam. The sliding piston-valve (Fig. 144) is preferable, because it is balanced and requires very little power to move it, but it does not give a sharp cut-off. The inability of either of these to alter the cut-off (*B*, Fig. 38) without also affecting the degree of compression, *DE*, makes the Meyer cut-off valve (Fig. 124) a better controller. Separate eccentrics drive the main slide-valve and the rider-valve, the latter to control the point of cut-off only, while the main valve regulates the exhaust and admits steam when the rider uncovers its port.

The Corliss Valve consists of four rotary valves, each with a single function to perform. One admission rotary valve and one exhaust rotary valve serve for each end of the cylinder. They are all moved by rods from a wrist-plate, the admission-valve being provided with a releasing trip-gear to make a sharp cut-off (Fig. 45).

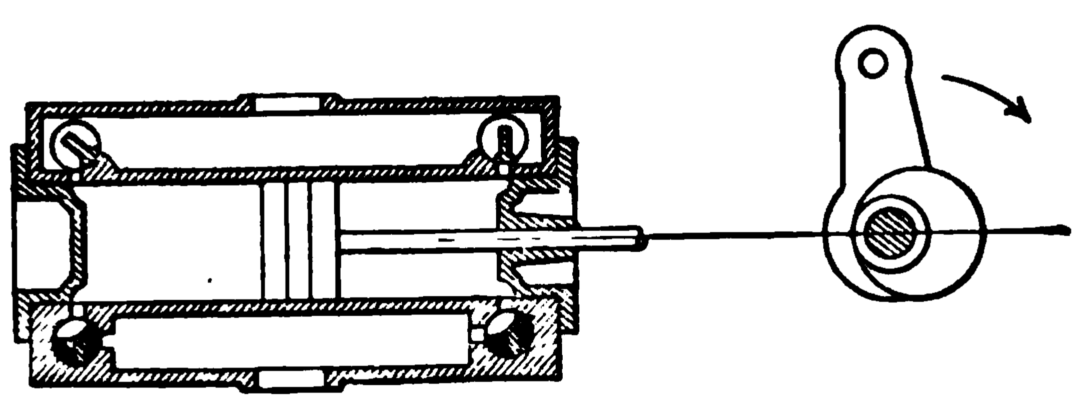

FIG. 45.—Section of Corliss Engine, showing valves in position for admitting steam at the left.

Fig. 126 illustrates the Corliss engine and valve in section on the left-hand air-cylinder.

The Reversing-link.—The reversal of the direction of motion of the engine is accomplished by one of two methods: engines supplied with shaft-governors may be reversed in direction by changing the position of an eccentric on the shaft and thus moving the valve to correspond; and slow-speed engines by the

Stevenson link motion with two eccentrics (Fig. 46). Motion is imparted to the steam-valve by a rod connected to an eccentric, on the crank-shaft. The eccentric is placed at an angle of something more than 90° in advance of the crank-pin position. When a reversal of motion is desired the single eccentric is turned to an angle of more than 90° in advance on the other side of the crank-pin. The Stevenson link arc has each end connected with an eccentric. When the arc is raised or lowered till the block to which the valve is attached is at the extreme end of the arc, the valve will have a maximum travel. In position *a*

FIG. 46.—The Reversing Link.

(Fig. 46) the block gives a forward motion, and when raised to the position *b* a backward motion, to the valve. In its central position the block is acted upon by both eccentrics and only a slight motion is imparted to the valve.

The Condenser is a receiver fitted with tubes of copper through which the steam may flow from engine to a tank or a pump before delivery into the boiler. Around the tubes large volumes of cold water circulate, condensing the steam and creating in the tubes a vacuum, which is further maintained by the air-pump. The vacuum obtained in practice is about 25 inches of mercury column below the atmosphere, or an absolute pressure of 2.5 lbs. per square inch. This addition to a plant economizes power by reducing the back pressure, increasing correspondingly the net effective pressure, as may be seen by comparing the indicator cards (Figs. 38 and 47), by nearly 15 lbs. per square inch. This is the quickest and simplest means of

increasing the power of a given engine and boiler plant. The saving in steam consumption is indicated for Corliss engines in a later table.

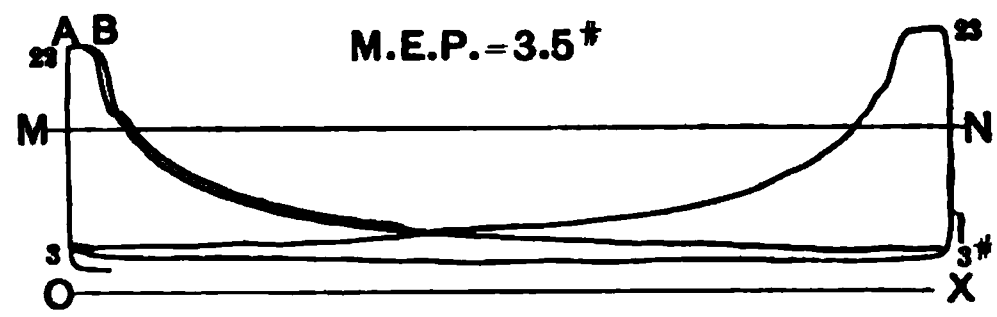

FIG. 47.—Indicator Diagram from a Condensing Engine or the Low-pressure Cylinder of a Compound Engine.

For every 100 feet of piston speed the condenser will save 1.03 H.P. in an engine whose cylinder is 6 inches in diameter; with a 10-inch cylinder, 2.86 H.P. is saved per 100 feet of veocity; with a 16-inch cylinder, 7.31 H.P.; and for 20 inches diameter, 11.42 H.P. Thus an engine with 10″×16″ cylinder making 80 r.p.m. (213.33 feet piston speed) saves 6.1 H.P.

Influence of Clearance.—The clearance space at each end of a cylinder prevents the piston striking the heads and provides a space in which steam accumulates to form a cushion and assist in the reversal of piston movement. A large clearance is not economical. Hence high-speed and slide-valve engines are less economical than the Corliss. A large clearance implies a small degree of compression in the cushion and less back pressure.

The most economical mean effective pressure is that corresponding to a minimum steam consumption. When the steam supply is cut off at a point of stroke that will permit it to expand to the pressure of the condenser, a complete expansion is obtained. The more complete the expansion the less will be the mean effective pressure. With a low M.E.P. a higher economy is attained than from a similar engine with a high M.E.P. A low pressure means large cylinders and excessive condensation. The degree of expansion and the number of cylinders are fixed by the ratio of the allowable boiler pressure to that of the exhaust into condenser or atmosphere.

An engine with a fixed load is operating at its maximum efficiency when driving its calculated load. If the load varies, the

engine must have a capacity for the maximum demand. If this load is casual and seldom demanded, the engine may be then forced. Its size will be determined by the demand of the average load, in supplying which it may attain its highest efficiency.

With a single-valve simple engine the most economical M.E.P. is about one half of the boiler pressure, which would represent a steam consumption of about 28 lbs. and a M.E.P. of 50 lbs. per square inch. A simple Corliss has its maximum economy at a M.E.P. of 35 lbs. pressure and a Corliss compound at 26 lbs. The steam consumption would be 25.3 and 19 lbs. respectively.

Selection of Engine.—Generally speaking, any expenditure for a plant is warranted which will show a saving more than equal to the interest on the expenditure, the depreciation and repairs of the plant. But this balance is not easily obtained. An economic engine is one with a low M.E.P., large cylinders, and costly attachments. But its initial cost may be prohibitive at the time of its installation. If the fuel is cheap and the amount of repairs not known, a high-pressure plant may not be desirable, particularly as its piping system usually gives trouble and the wear and leakages are large. Nevertheless the saving resulting from its use generally more than offsets the increased cost. The smaller boiler capacity and the diminishment of its cost should be considered with the other items of saving.

The Speed of Engines.—Engines are classified according to their piston speed, as slow-speed, averaging 100 feet per minute, and high-speed, making nearly 900 feet of travel per minute. This value is the product of the number of revolutions and the piston stroke—two independent variables.

The Corliss engine is limited to about 60 r.p.m., and electric engines for lighting, etc., make about 400 revolutions. For mining purposes the rate rarely exceeds 100 r.p.m. The high rotary speed is not economical, because of the large clearance at each end of the cylinder. The excellent workmanship and the great care which the high-speed engines require make their first cost and subsequent maintenance large. They have less steam condensation than slow engines.

The work of the piston is equal to the continued product of the mean effective pressure upon it, the stroke in feet, its area in square inches, and the number of strokes If the speed is fixed for a given engine, the area may be designed proportionately to s, the mean effective pressure being altered at will (by changing the initial boiler pressure or the cut-off).

The Horse-power of an Engine.—The *theoretical* horse-power is the product of the work of the piston and the number of strokes divided by 33,000. The area of the piston-rod on the crank side is neglected, as also the losses from friction of the engine parts, steam condensation in the cylinder, clearance, and compression. The value for the effective pressure, p, is obtained from the formula on a previous page.

The *indicated* horse-power (I.H.P.) is the actual work of the steam as revealed by the indicator card; the mean effective pressure is obtained (Fig. 39) by the card. It is less than p, obtained above by a coefficient, E, whose value is less than unity.

The "*brake*" horse-power (B.H.P.) is a certain fractional part, m, of the indicated, which allows for the mechanical losses and for intermittence in running. The value for the modulus varies from 0.60 in a very poor engine to 0.90 in one in good working order. A fair average for the mechanical efficiency, m, of the engine ratio between the brake horse-power and the indicator horse-power is 0.8.

Let s = the length of the piston stroke in inches;
k = the diameter of cylinders in inches;
N = the number of revolutions per minute;
$0.166Ns$ = the piston speed, feet per minute;
c = the number of cylinders;
f = the coefficient of friction;
p = mean effective steam pressure, pounds per square inch;
P = absolute steam pressure entering the cylinders;
r = ratio of expansion or per cent of cut-off;
B = absolute back pressure of the steam during exhaust;
H = horse-power developed by the engine.

Then

$$H = (0.000003966csk^2N)pm.$$

In the following table are calculated, for certain sizes of cylinders and given piston speeds, the values for the parenthetical expression.

THE HORSE-POWER FOR EACH POUND OF MEAN EFFECTIVE PRESSURE PER SQUARE INCH OF PISTON AREA.

The value for $0.000003966sk^2N$.

Diameter of Cylinder.	Average Speed of Piston in Feet per Minute.									
	240	300	350	400	450	500	550	600	650	750
4	.091	.114	.133	.162	.171					
5	.144	.18	.21	.24	.27					
6	.205	.256	.299	.342	.385	.428	.471			
7	.279	.348	.408	.466	.524	.583	.641			
8	.365	.456	.532	.608	.685	.761	.837	.912		
9	.462	.577	.674	.770	.866	.963	1.059	1.154		
10	.571	.714	.833	.952	1.071	1.190	1.309	1.428	1.547	1.785
11	.691	.864	1.008	1.152	1.296	1.44	1.584	1.728	1.872	2.160
12	.820	1.025	1.195	1.366	1.540	1.708	1.880	2.050	2.222	2.564
14	1.119	1.398	1.631	1.864	2.097	2.331	2.564	2.797	3.029	3.495
15	1.285	1.606	1.873	2.131	2.409	2.677	2.945	3.212	3.479	4.004
16	1.461	1.827	2.131	2.436	2.741	3.045	3.349	3.654	3.958	4.567
18	1.849	2.312	2.697	3.083	3.468	3.854	4.239	4.624	5.009	5.78
20	2.292	2.855	3.331	3.807	4.285	4.759	5.234	5.731	6.186	7.138
23	3.021	3.776	4.104	5.035	5.664	6.294	6.923	7.552	8.181	9.44

EXAMPLES.—1. Required the horse-power of an engine 6″×10″ with a piston speed of 300 feet per minute (180 revolutions) with M.E.P. of 40 lbs. According to the table, for $k=6$ inches and the piston speed of 300 feet, a, constant of 0.256 is found. For 40 lbs. pressure and $m=0.80$ the horse-power, $H=0.256\times40\times0.80=8.19$.

2. Required the horse-power of a duplex engine 10″×14.4″ making 100 r.p.m. and having a cut-off of ¼ with 7 per cent clearance and initial admission pressure of 100 lbs. absolute.

Let $m=0.80$ and back pressure 18 lbs. according to table on page 105. Then $p=0.6861(100)-18=50.64$. $C=0.6861$.

According to the above table the constant for 10-inch cylinder and 240 feet piston speed is 0.571. Whence $H=0.571\times50.64\times0.80=23.13$ for each cylinder and 46.16 for the duplex engine.

Usually there is an approximate ratio between the diameter, k, and the length, s, of a cylinder. In a long-stroke engine $s=2k$ to $2.5k$. A short-stroke engine has a ratio $s=k$ to $s=1.2k$.

The Steam Consumption of an Engine.—A one-horse-power engine exercised through one hour, equals 1,980,000 ft.-lbs.; one B.T.U. = 778 ft. lbs.; hence one horse-power hour corresponds to 2545 B.T.U. A pound of steam represents about 1100 B.T.U. above feed-water temperature. Hence one horse-power should be obtained during one hour from 2.3 lbs. of steam. The very best on record is 12.2 lbs. per I.H.P., and for a stationary engine of the Corliss type, compound, about 18 lbs. Hence all steam consumption above 12.2 lbs. may be regarded as preventable waste.

The following table shows the consumption of steam per hourly horse-power with non-condensing and condensing engines at various rates of cut-off and different boiler pressures. An allowance is here made for ordinary cylinder losses. The slide-valve engine would show a steam consumption under similar conditions of nearly twice those indicated in the table.

In a slide-valve engine 22″×28″ at 118 r.p.m. with a boiler pressure of 110 lbs. absolute, clearance at 8 per cent, a mean effective pressure of 43.1 lbs. is obtained. By calculation from the indicator diagram the steam consumption was 0.64703 lbs. per stroke. The same size of Corliss engine with a $\frac{1}{8}$ cut-off and 3 per cent clearance used 0.32635 lbs. of steam per stroke. Each engine making 14,160 strokes per hour, the steam consumption will be, in the two cases, respectively 33.5 and 17.08 lbs. per horse-power per hour. The Corliss therefore saves nearly one-half the steam consumption of the slide-valve engine. A 100-H.P. Corliss will annually save 150 tons of coal over a slide-valve engine.

Setting the Engine.—A foundation is prepared for the engine before its arrival. Two accurately constructed templates, ordered from the manufacturers, are laid in proper position with bolts connecting them long enough to reach from the bottom to the top of the proposed masonry foundation. Upon the lower one a foundation of concrete or brick is built, without disturbing the bolts, to the level of the bed-plate, which is finished and adjusted ready for the engine. The upper template may be removed and the engine at once dropped into position without delay. Timber foundations are used only for second-motion haulage-engines

THE MEAN EFFECTIVE PRESSURES AND TERMINAL PRESSURES AND STEAM CONSUMPTION IN CORLISS NON-CONDENSING AND CONDENSING ENGINES.

Initial Gauge Pressure.	1/6 Cut-off. M.E.P. N. C.	M.E.P. C.	Terminal Pressure.	Water Consumption. N. C.	Water Consumption. C.	1/4 Cut-off. M.E.P. N. C.	M.E.P. C.	Terminal Pressure.	Water Consumption. N. C.	Water Consumption. C.
60....	24.24	36.24	16.08	27.6	20.5	29.56	41.56	19.77	27.3	21.2
65....	26.93	38.93	17.15	26.5	20.3	32.61	44.61	21.09	26.4	21.0
70....	29.63	41.63	18.23	25.7	20.1	35.67	47.67	22.41	25.8	20.8
75....	32.32	44.32	19.31	24.9	19.9	38.72	50.72	23.73	25.2	20.6
80....	35.02	47.02	20.39	24.3	19.8	41.78	53.78	25.05	24.8	20.4
80....	37.73	49.71	21.46	23.7	19.7	44.83	56.83	26.37	24.3	20.3
90....	40.41	52.41	22.54	23.4	19.5	47.89	59.89	27.69	24.0	20.3
95....	43.10	55.10	23.62	23.0	19.4	50.94	62.94	29.01	23.6	20.2
100....	45.80	57.80	24.70	22.6	19.4	54.00	66.00	30.33	23.1	20.1
	3/10 Cut-off.					35/100 Cut-off.				
60...	34.21	46.24	23.37	27.4	22.0	38.18	50.68	27.04	28.3	22.8
65....	37.61	49.61	24.94	26.7	21.9	41.80	53.80	28.85	27.5	22.6
70....	40.98	52.98	26.51	26.1	21.7	45.42	57.42	30.66	26.8	22.5
75....	44.35	56.35	28.07	25.6	21.5	49.05	61.05	32.47	26.2	22.4
80....	47.72	59.72	29.64	25.1	21.3	52.68	64.68	34.28	25.7	22.3
85....	51.09	49.71	31.20	24.6	21.2	56.31	68.31	36.09	25.3	22.2
90....	54.46	66.46	32.77	24.2	21.0	59.94	71.94	37.90	25.0	22.2
95....	57.83	69.83	34.33	23.9	20.9	63.57	75.57	39.71	24.7	22.0
100....	61.20	73.20	35.90	23.7	20.8	67.20	79.20	41.52	24.5	21.9
	4/10 Cut-off.					1/2 Cut-off.				
60....	41.68	53.68	30.66	28.8	23.6	47.50	56.50	37.98	29.4	25.2
65....	45.54	57.54	32.71	28.2	23.4	51.75	63.75	40.52	28.8	25.0
70. ..	49.41	61.41	34.77	27.6	23.2	56.00	68.00	43.07	28.4	24.8
75. ..	53.27	65.27	36.82	27.1	23.0	60.25	72.25	45.61	27.9	24.7
80....	57.14	69.14	38.88	26.7	22.9	64.50	76.50	48.16	27.6	24.6
85....	61.00	73.00	40.93	26.1	22.8	68.75	80.75	50.70	27.2	24.5
90....	64.87	76.87	42.99	25.8	22.6	73.00	85.00	53.25	27.0	24.4
95....	68.73	80.73	45.04	25.5	22.5	77.25	89.25	55.79	26.8	24.2
100....	72.60	84.60	47.10	25.2	22.4	81.50	93.50	58.34	26.6	24.1

These, reduced to coal consumption at the rate of 10 lbs. of steam per pound of coal, can be at once referred to the coal-pile.

The Rotary Effort.—The "horse-power" of the engine assumes a uniform effort of the engine against the resistance, but owing to varying angles occupied by the crank-arm, which receives the thrust from the connecting-rod, the rotary effort upon a

crank-pin varies from one dead-center to a maximum and thence diminishes to a minimum during each half of the revolution or each stroke. Owing to this variation in effort the cylinders are usually duplicated side by side, their pistons operating upon their respective cranks 90° apart. This enables the piston of one cylinder at the middle of its stroke to assist the piston starting its stroke.

The governors, discussed above, check variation of speed from stroke to stroke but not during each single stroke. The variations in the rate of rotary effort upon the crank-pin must be kept within reasonable limits by use of a fly-wheel. There is a varying pressure on the piston due to expansion, a varying thrust upon the connecting-rod due to its angularity, and a varying tangential effort upon the crank-pin due to the angular position of the crank-arm. These variations cause a fluctuation in the speed of the crank-pin, the crank-shaft, and of the hoister-drum which is attached to it. The shorter the connecting-rod the more nearly equal are the forces throughout the stroke and in each half-revolution. A diagram drawn to represent the tangential effort upon the crank-pin by vertical lines and the rectilinear motion of the crank-pin by the horizontal distances; the curve shows the varying efforts, or torque, of the engine. If diagrams be prepared representing the rotary effort of each piston of a duplex engine and be superposed upon one another, at the same time advanced 90° to represent the difference in the angle of the crank-arms, the sum of the ordinates at any one point represents the aggregate rotary effort of two pistons upon the crank-shaft.

An engine must be capable of starting from any position of its piston, or of its corresponding crank-pin, and have at that instant a rotary effort in excess of the resistance on the crank-arm.

The Steam-turbine may be employed for purposes of a motor engine where a high rate of revolution is desired, as, for example, with a centrifugal pump or an electric generator. For these purposes it is particularly adapted.

The steam-turbine is a wheel fitted at its circumference with numerous blades receiving the impact of a jet of steam flowing against them at a high velocity. The turbines may be classified according to the path of the steam, which may be radial, outward or inward, parallel flow or spiral flow. The turbine utilizing the steam which flows in a direction at a constant distance from the axis of the wheel is the one most highly developed at the present time. Of these the De Laval single wheel and the Parsons, Curtis, and Rateau multiple wheels are the types.

The De Laval Wheel.—In this a jet of steam is driven through a nozzle, *D* (Fig. 48), where it expands to the atmospheric pressure, thus creating a high velocity of flow whose energy is imparted to the blades of the wheel producing a peripheral speed of 30,000 feet per minute. This wheel, being 8″ or more in diameter, makes 15,000 or more revolutions per minute. This rate being too high for motors, a reduction gear is employed. Pumps and electric generators are attached to the shaft, making 1000 or more revolutions. In Fig. 48 are shown also details of its construction.

The Parsons Steam-turbine.—This is a series of rotary discs fitted with blades around their circumference (Fig. 49). Alternating with the discs are rings of fixed blades projected inward from the casing. The steam is discharged from a nozzle, thence to the fixed guide-blade, which deflects it to the rotary blade. This, after absorbing some of the power, again discharges the fluid to the next guide-blade, etc. Thirty or more such passages are encountered before exhaust, and each one of the 16,000 rotary blades receives a small amount of pressure, the aggregate of which produces the effective rotation of the shaft. The discs are fitted on the one axle and revolve together, making 400 revolutions per minute.

The Curtis and Rateau types of wheels are very similar, but occupy intermediate positions between those already mentioned, as to the method of utilizing the energy of the steam.

The Advantages of the Steam-turbine.—It is a rotary machine, requiring very small floor space and little or no foundation. It

is much lighter in weight than is a steam-engine delivering the same amount of power. These steam-turbines are employed at pressures not to exceed 150 lbs. gauge. The steam should be dry and to a moderate degree is superheated before being delivered to the turbine. Condensers may be added to increase their power.

Its steam consumption is about the same as that of a steam-engine of equal power. There being no internal lubrication,

FIG. 48.—De Laval Wheel Details.

the steam does not come into contact with a running part and its condensed exhaust can be returned to the boiler free from oil. This is a merit of great importance where the supply of feed-water is chemically bad. There are no complicated valves to give occasion for injury, and no pistons to leak. All bearings are easily lubricated. The only wearing parts are the nozzles and the blades. Turbines can be had in power from those of 10 horse-power up to one of 7400 I.H.P.

In point of efficiency and commercial utility it has the field, which will soon be undisputed. Its thermodynamic efficiency is equalled only by the gas- or oil-engine.

FIG. 49.—Parsons Turbine.

Oil-engines.—When the distances are too remote for economical admission of steam or compressed air through the pipes or of electricity by wires to pumps and fans, independent oil- or gas-engines are employed. These are self-contained and are placed on the same foundation with the machine to be driven, and may even be portable on a truck. These engines require little attention and have a high efficiency in addition to the advantage of portability. The motor fluid is gasoline, naphtha, or alcohol, which is held in a supply-tank from which, by means of a small pump attached to the engine, it is conducted to the working cylinder after having been atomized or vaporized. It is then mixed with air in proportions regulated by a suitable device. This vaporous mixture is drawn into the cylinder during the first stroke of the piston, compressed into a large clearance space in the second stroke, at the end of which the mixture is ignited and expanded, the explosive force of which drives the piston forward on its third and outward stroke. On the return or fourth stroke the products of combustion are expelled into the atmosphere. The operation is then repeated, there being but one working stroke out of each four strokes or every two revolutions of the fly-wheel. This constitutes what is known as the four-cycle engine (Fig. 50). Two or more such cylinders may be attached on the same shaft, with their cranks at various angles to produce uniform rotary effort. A very heavy fly-wheel is essential to carry the engine over the three non-working strokes. It is started by hand and thereafter requires little attention. Its thermodynamic efficiency exceeds that of the steam-engine, and it is useful where the power must be continuous, as for electric generation or pumps. Fig. 147 illustrates one of these internal-combustion engines of the two-cycle type, the compression being conducted outside of the cylinder.

It is essential that the amount of air be chemically perfect for complete combustion, when the initial pressure attained will be 200 or 300 lbs. per square inch. The temperature of the explosion is over 1000° F. The cylinder is cooled by the water circulating in a jacket, and the exhaust-gases are delivered into

the stack. The former absorbs from 40 to 50 per cent of the heat developed in the combustion, and the exhaust carries off 30 to 40 per cent more, leaving about 20 per cent for effective work. The oil-consumption per horse-power hour is from $\frac{1}{4}$ to $\frac{1}{3}$ of a gallon.

Ignition may be attained by an electric spark or a tube kept red-hot by a flame during the entire running. The efficiency of

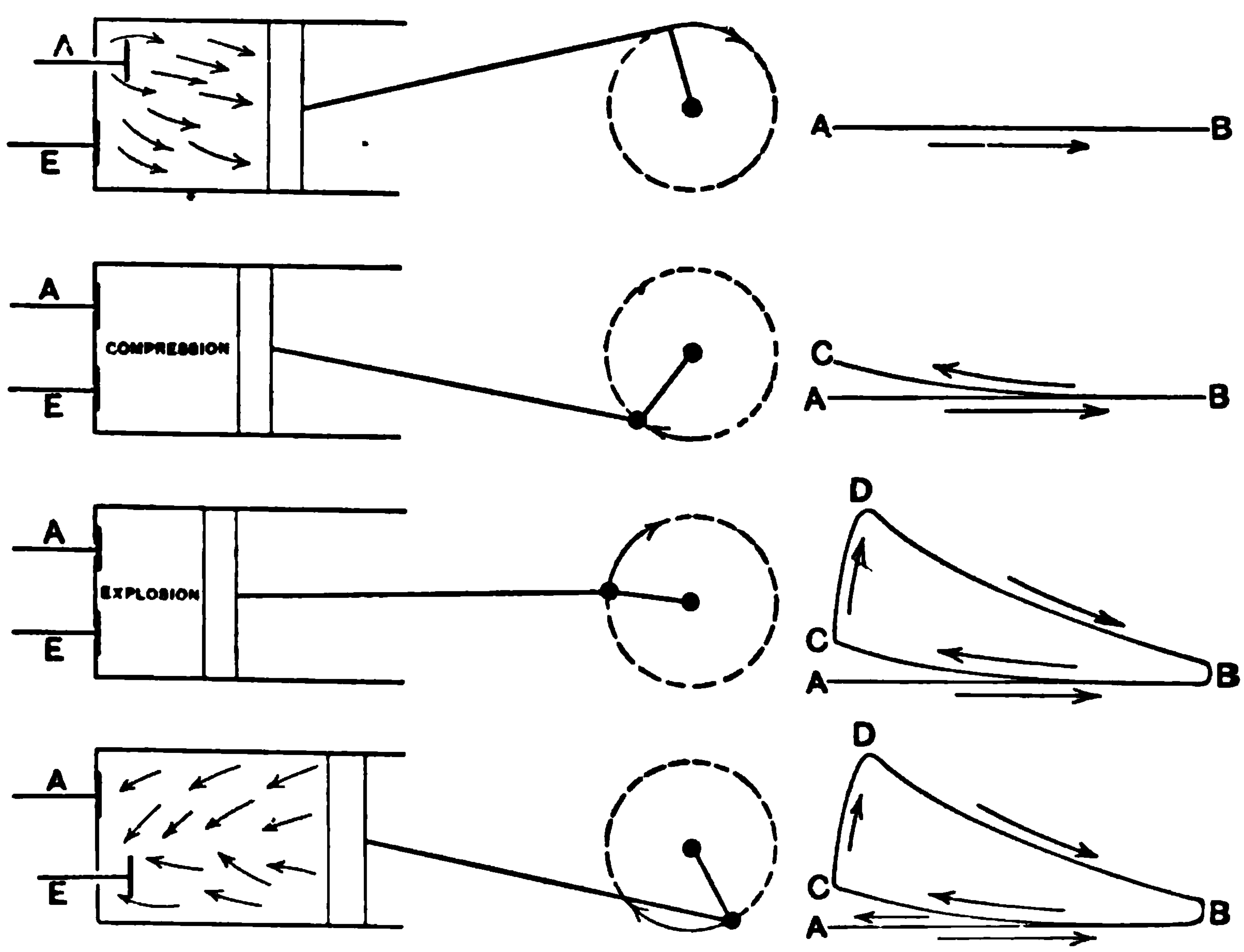

FIG. 50.—The Four Cycles of a Gas- or Oil-engine.

the oil-engine increases with the increase of the degree of compression at the end of the second stroke. It follows therefore that the Diesel motor, which raises the compression to 700 lbs. per square inch—nearly the point of ignition of the gases—gives the highest results (Fig. 147). One I.H.P. is obtained from the consumption of 0.3 lb. oil.

The regulation of these internal-combustion engines is obtained by a governor, which produces a missfire when the engine is underloaded.

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CHAPTER V.

HOISTING MACHINERY.

Hoisting Machinery.—The various mechanisms used for hoisting minerals in the mines are the windlass, the whim, the whip, and the engine. The first-named is employed for moderate depths of mine and in auxiliary underground openings where neither the weight to be hoisted nor its distance is very great. The whim and whip usually represent the next stage in the development of a metalliferous mine, when the limit of the former hand methods has been exceeded. These are driven by a horse, or team of horses, and are employed for depths not exceeding 200 feet. The steam-engine is the motor for all mines having a large output or a great depth of shaft, and even for a depth of but 100 feet is still more economical than the simpler methods mentioned above.

Windlasses are employed for shafts of moderate depth and for winzes underground where manual labor is depended upon for hoisting purposes. The amount that can be raised by two men 100 feet in eight hours cannot exceed 4 tons, allowing for delays, etc. The windlass is a barrel, the wooden cylinder, 6 inches to 10 inches in diameter, long enough to reach across the shaft, raising by its axle on uprights, and operated at each end by cranks 15 inches long set at right angles to each other. Winches and crabs or windlasses, in iron, can also be had in every possible combination, but the simpler the machine the less is the friction, and the more acceptable it is. Each additional gearing involves a large percentage of loss, and there is little to spare from the average man-power of 5300 ft.-lbs. per minute. Windlasses

are, however, useful for incidental purposes in handling heavy pieces of timber, machinery, and pump-pipes. A given power can only equal a certain product of weight and velocity; for an increase of speed the weight must be proportionately diminished. With a 15-inch crank-arm, 12 revolutions per minute, coil 25 feet of rope on an 8-inch barrel, and with this speed of hoist the greatest load that may be moved under the circumstances by an average laborer is 214 lbs. Friction and stiffness of the rope will reduce this. A 150-lb. load can be raised at a speed of only 35 feet per minute.

Balancing the Loads.—The lengthened windlass barrel receives several coils of rope, from the ends of which two buckets are suspended, the one falling to balance that rising. The length of rope is a few coils only greater than the depth of the shaft. At the start the weight to be hoisted is only the contents of the tub plus the rope. This weight diminishes in rising, till at the top it is the contents minus the rope. This does not, however, obviate the great stress from inertia which arises at the moment of starting. For this reason single or double conical barrels are used, on which rope is coiled in such manner that the empty bucket is hung from the larger diameter, while the rope from the loaded tub at the bottom is wrapped around the smaller end. The tubs balance each other, but the empty acts with a greater leverage than the loaded tub, and thus assists the power in overcoming the inertia. After the hoisting is under way, the empty and its lengthening rope uncoils with diminishing leverage, while the load with its shortening rope gradually winds on a larger diameter of the cone The buckets do not meet in the middle of the shaft, but in the middle of the number of revolutions of the barrel. In any event the pitch of the cone must be calculated for the given conditions of depth and load, otherwise its advantage is manifest only at the start. When properly constructed the conical drum is to be recommended. Not many attempts are made to apply this mechanism to hand hoisting, but it is rapidly coming into favor with horse- and engine-power, notwithstanding that it is dearer than cylindrical drums.

Manual labor is manifestly too expensive to be regarded as any but a temporary expedient for hoisting.

Whims.—When the height of he hoist exceeds 60 feet or the output 5 tons per shift, horse-power is employed to advantage. The average horse develops an effort of 135 lbs. when walking at a speed of 180 feet per minute. This is much below the theoretical value fixed for a horse-power, yet it represents the results of many tests upon the energy of the animal, which will raise 9 tons 150 feet per day.

For a slightly greater quantity two horses are used. But when a larger quantity is to be handled, a more efficient and economic power is employed. The utilization of horse-power is but an intermediate step in the history of many mines. Where water is scarce, fuel dear, and the transportation of machinery difficult, the horse-whim serves temporary ends as a simple and tolerably satisfactory hoister.

The invariable arrangement is a wheel-and-axle machine, consisting of a drum and driving-beam to which the horse is harnessed. Two sticks 6″×6″ and 9 feet long are mortised together at right angles to each other, with four 4-inch planks trimmed to the quadrant of a circle. These are held a foot or two apart by studs, and to them 3-inch plank staves are spiked to form the barrel, which, though upheld on the axle, turns freely about it. The axle is a round 2-inch rod stepped in a stone or iron block and held at the top in an iron socket on the span-beam. The latter is 10 inches square, 36 feet long, supported on legs mortised to it and braced A square iron axle-rod fastened to the drum and turning with it is often seen, but is not so good as the free axle. The entire frame can be built for $100. The drum may be above or below the driving-beam (Fig. 51).

A derrick frame is necessary over the shaft at a height sufficient for convenience of handling the hoisted tub. The hoist-rope passes over a sheave at the top and to the drum if set up high over the driving-pole, or under another pulley at its base if the drum is close to the ground. The latter arrangement is

FIG. 51.—A Whim.

cheaper to build, but is wasteful of power. For lowering the tub (Fig. 51), a lever is in reach of the driver, by which the driving-beam may be disengaged from the drum and the tub lowered by its own weight, uncoiling the rope from the drum. A band-brake $3'' \times \frac{1}{4}''$ regulates the speed. The brake must be set so as to work with the motion, not opposite to it; and the brake force exerted to produce larger tension in the driving, not in the slack, portion of the band. The length of the driving-beam and the diameter of the drum may be altered at will, but the ratio between them is also the ratio of the speed of the horse to that of the hoist.

If circumstances permit, two ropes may be operated from the same drum, one ascending and the other descending Conical drums may also be employed. Iron-framed whims are on sale, in which a drum is horizontal and turned by a bevel-gear on its axle, fitting to another at the central end of the drive-beam. While convenient and easy to erect, the introduction of the bevel-gear involves additional friction.

The plane of the derrick-pulley should be tangent to the drum, and the latter far enough away that the rope may coil and unwind freely without chafing on its adjoining coil. This is accomplished by arranging the point of departure of the rope from the pulley at the same height as the central coil. Where the full and empty tubs are simultaneously operated, this can be attained only approximately. If greater nicety is desired, the pulley slides on an inclined plane as the coiling or uncoiling proceeds, or else the drum shifts its position by turning on a screw-thread. No lateral motion is allowed to the sheave, since the rope must occupy a central position in its own hoisting compartment.

The Hoisting-engine.—A hoister is a simple engine attached to a drum on which a rope is wound The combinations are numerous, from that of the upright boiler and engine on the same base, to the vertical condensing, compound engine connected with drums on separate foundations. Generally the cylinders are horizontal. The choice between horizontal and

Fig. 52.—Duplex Corliss Hoister showing Disc-cage Indicators.

upright engines is chiefly one of space. The horizontal engine is the cheaper, the simpler, the easier to inspect, and the easier to repair. Outside of the advantage of requiring less space, the upright engine has the advantage of less wear on the cylinder, and a more direct strain upon the foundations. The types are few, from the slow haulage-engines for tail-rope to the high-speed hoists with large drums Most hoisters are operated intermittently; only in a few instances are they continuous. The engine must be simple, safe, under complete control, capable of quickly attaining full speed and being accurately stopped.

The selection of the type will largely depend upon the speed desired for hoisting and the choice of valves, condensers, ratio of expansion, compound or simple engine, and the comparative cost and maintenance. Frequently the question of economy receives the last consideration, simplicity of construction and safety of operation being of prime importance.

First- and Second-motion Hoisters.—The communication of motion from piston to the drum is direct or secondary and may or may not be provided with a reversing motion. In the former class the piston is coupled directly to the drum through the connecting-rod. The load is hoisted, held, or lowered by steam at the will of the engineer. High speed is possible with direct-acting engine (Figs. 52, 57 and 58.)

Second-motion Engines.—All engines transmitting the power to the drum through one or two pinion wheels or a friction-clutch are second-motion engines. These engines are slower but under greater control than the direct-acting engines, besides offering less risk to overwinding. The pinion-wheels may be toothed wheels gearing with others on the drum-shaft, or friction-wheels bearing against those on the drum-shaft (Fig. 53).

The ratio of this gear varies between 4 to 1 and 7 to 1, the latter ratio being for very slow speed. Some quarry engines are provided with a double set of gear-wheels for two speeds (Fig. 54).

The geared engine is intended for slow hoisting, shallow depths, and small outputs. The direct-acting is employed for

FIG. 53.—A Second-motion, Duplex Corliss, Conical-drum Hoister, showing Mechanism of Cage Indicator.

FIG. 54. Geared Haulage-engine

large outputs and with a hoisting speed of never less than 500 feet per minute. The limit to the size of the gear-wheels is determined by the peripheral speed of the teeth, which should not exceed 1000 feet per minute.

Second-motion engines may be operated with a short cut-off, which is not always possible with the direct-acting tyye. More. over, their repairs are greater and the operation easier. A properly designed second-motion engine may be employed economically on steep slopes, and occasionally for short shafts where the output is large. A pair of 16″×30″ cylinders on a three-foot drum had an output of 1413 tons from a shaft 370 feet deep in six hours' time.

Internal Gear Connection.—Frequently drums are fitted with pinion gearing into an internal rack on the drum. This can be slipped out at will, leaving the free drum to pay out the rope until the proper depth is reached, when the pinion is returned to place, and the drum is ready for hoisting from that depth. This arrangement is in ended for the variable lengths of the hoist.

Friction-engines have a pinion-wheel on the driving-shaft of papier maché bearing against an iron surface on the drum-shaft. The surfaces are kept dry and free from water, steam, or other substances which reduce the coefficient of friction and thus require an excessive pressure to move the requisite load. Friction-engines deliver the power to the drum more quietly than toothed-gear or direct-connected engines, and therefore produce less shock upon the rope. Sometimes the engine is in continual rotation with its pinion-shaft, and hoisting is effected by bringing the drum with its shaft and gearing surface against the pinion. For lowering, the drum is released from contact with its pinion and revolves freely, being controlled only by the brake manipulated by the engineer. The risk of slipping and the inability of the engineer to apply the requisite pressure makes the friction-engine unsafe.

V **Friction-wheels.**—The surfaces of the drum-band and the driving pinion are grooved circumferentially to give a tight grip between them without excessive pressure on the bearings. These

FIG. 55.—Internal-friction-clutch Hoisting-drum.

wheels are of cast iron. The irregular wear of the wedges, however, makes them unreliable in time.

Friction-clutches.—These engage the drum externally or internally. They are quick, noiseless, and give little shock. A number of blocks attached to a band furnish the frictional surface, and are directly attached to the revolving shaft and in continual rotation about the drum. They engage the outside of the drum when forced by a two-armed driver operated by bell-crank levers and revolve the drum (Fig. 56). A grooved sleeve or cone moving along the shaft may also cause a shortening of the arms which bring the band down on the drum and impart their motion to it. For lowering, the band is released and the drum revolves while held in check by the brake.

One form of internal friction consists of driving-bands with four arms keyed to the drum-shaft and in continuous motion. The drum is free on this shaft and will hoist as soon as the band is expanded to contact with its rim, by sliding the sleeve upon its shaft and engaging its arms against its inside rim (Fig. 55).

The Brake.—This is either an iron band with blocks of hard wood on end, or a V-grooved wheel bearing on a corresponding surface on the drum. The first is less troublesome and safer. It is suitably applied by a simple lever, by a hand-wheel and worm-screw, or an auxiliary steam-piston, according to the size of the hoist. Some large engines require a small regulating-engine to stop and start them.

The Unbalanced Load on the Engine.—The resistance which an engine must overcome will include the unbalanced weights, the friction and the bending stress of the rope. These having been determined, the size and the weight of the rope become known.

The maximum speed of hoist is limited by the equipment of the shaft, and the minimum by the hourly output desired. Owing to the diminishing weight of the rope as the hoisting progresses and the unequal work thrown upon the engine, a constant speed of hoisting cannot be maintained. The engine, from the beginning, speeds up with the reducing load, while the acceleration increases almost to the end of the hoist.

Fig. 56.—External friction Band-hoister.

FIG. 57.—A Duplex Corliss Engine with Reversing Gear and Cage Indicators.

FIG. 58.—Slide-valve, Reversing-link-motion, Conical-drum Duplex Hoister.

Balancing the Dead Load.—The work of the engine divided between the live load and the dead load is unequal because of the diminishing influence of the rope during the hoist. This inequality can be partially reduced by balancing the weight of the rope. The cages and the cars carrying the load may be balanced by attaching to another rope on the same drum and engine a similar cage and car. The weights of the two, going in opposite directions balance one another, though the weight of the rope, its friction, and its bending stress are still unbalanced; so also, the weight of the mineral being hoisted.

The balancing of cage and car may be accomplished by the use of two cylindrical drums (Fig. 57), a conical or a fusee drum or a counterbalancing system.

Balancing by Conical Drum.—The method of balancing by the use of a cone or a fusee depends on the change of leverage of the two loads. The load is hoisted from the small diameter of the drum, while the descending rope is suspended from the larger diameter of the cone. As the hoisting proceeds, the ascending rope winds upon an increasing diameter, while the descending rope and cage are gradually operating from a shorter radius of the drum. Equalization is somewhat maintained and an easy start is thus made.

A plain conical drum, to be effective as a counterbalance, requires an angle so great as to be dangerous on account of the liability of he rope to slip. Any advantage to be derived from having a safe angle being trifling, it is really not worth the risk.

The cylindrical drum admits of extension of the hoisting depth but the conical or fusee drum is built for a specific condition of load, speed and depth.

The Conical Drum.—This is used when the rope is heavy and the economy of accurate counterbalance is clearly indicated as practicable and will warrant the increased cost. It is much larger than the cylindrical drum, even if its mean diameter is the same. The depth of shaft or the point of hoist must be fixed to maintain equalization with a given cone or fusee. Hoisting from several landings would not be economical without the

use of additional counterpoises. The conical drum is employed single or double (Figs. 53, 58, and 59).

In designing the single or double conical drum the horizontal distance, P, between the centres of the two consecutive grooves or coils, and the vertical distance, p, between them must be in such ratio that the inclination shall not exceed 30°; $p \div P$ must be less than 0.577 The minimum diameter, $2r$, of the smaller end is fixed by the rope, and the larger diameter, $2d$, is dependent upon it.

Let c = the circumference of rope circle $= 3.1416(2r+x)$;

$p' = 0.2830p$;

L = length of the drum between the first and last grooves.

The length can be determined when the number of grooves has been calculated from the conditions given.

$$n=\frac{\sqrt{(2c-p')^2+8p'D}-2c+p'}{2p'}.$$

FIG. 59.—A Double Conical Hoist.

The Fusee.—This drum is designed to give an equalization of the dead load, including the rope, at all points throughout the journey of the cage. It may be determined by the use of the following formula:

$$(RD+A)r=Ad=M.$$

After one revolution ($RD+A$ and the weight of one turn of rope at the small end) ρ must balance ($A+$ the weight of one turn of rope at the large end) ρ' and still equal $Ad=M$. For each revolution throughout the journey this balance must be maintained. This is satisfied by the following formula, in which Z is the total arc of revolution described by a point on the rope at the small end from the beginning to the end of the hoist:

$$Z\int_{\rho-d}^{\rho-r} = 2\pi n = \frac{\sqrt{\frac{M}{R^2}-r^4}}{2r^2} - \frac{\sqrt{\frac{M^2}{R^2}-d^4}}{2d^2}.$$

R is the weight of one foot of rope.

The curve of the drum is then constructed by substituting different values between r and d, which are the limits of ρ, placing the second member equal to $2\pi n$, and solving for n, the number of grooves. The various assumed values for ρ are radii of the curve at the various points along the axis which are at a distance from the initial point equal to the horizontal pitch, P, multiplied by n. The curve so plotted is the section of the drum which will fulfil the conditions, furnishing an equalization throughout the journey.

Let $D=2000$ feet, $R=3$ lbs., $A=4000$ lbs., $r=48$ inches; then $M=40{,}000$ ft.-lbs., and d becomes 10 feet. Solving the equation and equating with $2\pi n$, $n=55.8$ revolutions between the initial and final points of winding. Where $d=9$, n becomes 53.3 revolutions from the start; where $d=8$, $n=49.5$; where $d=7$, $n=44.4$; and where $d=5$, here have been 35.1 coils of rope.

The Reel.—This is a barrel with spider arms on either side, between which a flat steel rope is wound in consecutive layers. It may be so devised that the moment of resistance will be nearly constant. The thickness of the coiling rope may be so selected that it will increase the leverage of the load at the same rate as the weight of the load decreases, in which event the work of hoisting is uniform. If however, uniform hoisting cannot be obtained throughout the trip with a reasonable thickness of rope,

FIG. 60.—Plan of a Geared Reel Hoister, showing the Brake Connections.

the design must be carefully considered and the engine tested after being placed (Fig. 60).

The smallest allowable diameter of barrel is fixed by the rope, and the number of layers, n, of the rope is equal to $(d-r)\div t$. The diameter of the coil, $2d$, when the cage is at the top of he hoist may be ascertained by the use of the formula $Dt = 3.1416 (d^2 - r^2)$. Given t ($\frac{3}{8}''$ to $\frac{3}{4}''$), the value for d may be obtained; or, having decided upon d, t can be ascertained.

Counterbalance.—The balancing of the rope weight and its resistance is accomplished either by the varying weight of an attached counterpoise to the cages or drum, or by varying the leverage on the hoisting-shaft of a fixed weight. The Koepe and Whiting systems and the chain balance are of the first type, and that installed at the Camphausen mine is of the second type. This makes the moment of the static load during the hoist constant and equal to that of weight of mineral The best results are obtained from a counterbalance when the hoist is always from one and the same level

The Koepe System of Winding.—This form of counterpoise consists in extending a tail rope from the bottom of one cage, under a sheave at the bottom of the shaft, up to the floor of the other cage. The usual hoist-drum is replaced by a simple grooved sheave, Fig. 61. Not only is the weight of rope, cage, and car in each compartment constant and the work of the engine nearly uniform, but the comparative lightness of the sheaves and the absence of heavy coils revolving with the drum give a correspondingly less inertia, and hence require less engine power, than with the ordinary systems. The net load which can be carried on the rising cage cannot exceed the friction caused by the aggregate weights suspended from the upper sheave, or else it will slip. A positive grip must be taken on the rope by the driving mechanism to prevent its slip or creep on the sheaves. Winding the main rope on a pair of cylindrical drums, or wrapping it a few times over a pair of multiple-grooved sheaves, as is the European practice in power transmission, Chapter IX, would accomplish this. Should such a slip take place, the indicators would fail to give the correct location of the cage.

This system meets almost all the requirements of a perfect equilibrium, is highly efficient, insures against overwinding, decreases wear, and disposes of an enormously heavy drum by using a sheave instead. But the lower sheave blockades the bottom of the shaft. The tail-rope may be a discarded hoisting-rope,

FIG. 61.—The Koepe System of Winding.

for it has no work to do beyond supporting its own weight. The method is of limited application as to depth.

In a shaft 1260 feet deep a saving of 11 per cent in power is effected. With sheaves 8½ feet in diameter and 8-inch axle, main rope 175 lbs. per foot and tail-rope 1.55 lbs., cage and car 2100 lbs., load 2750 lbs., depth of shaft 2500 feet, and velocity

of hoist 2500 feet per minute, 250 horse-power are saved over that required to operate the same loads without this balance.

The Whiting system (Fig. 62), a development of the Koepe

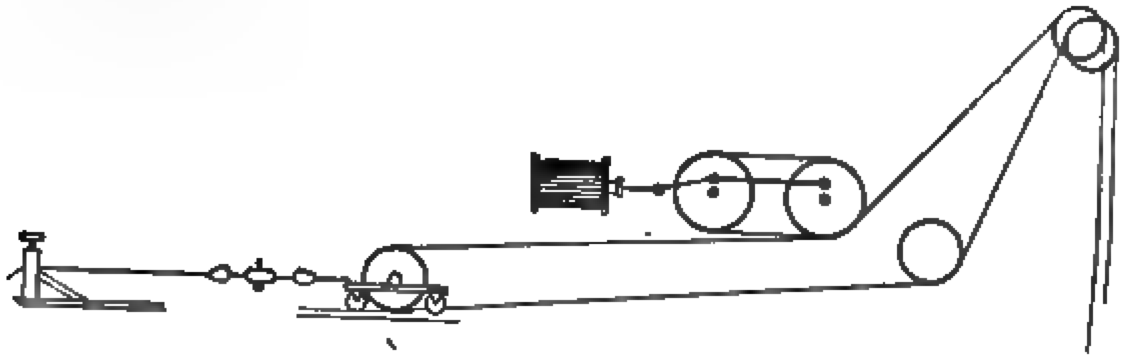


FIG. 62.—The Whiting System of Balance.

system, uses two sheaves and dispenses with the heavy coils of rope revolving with the drum.

Chain Counterbalance.—Another form of counterpoise is a heavy chain wound on a secondary drum on the main shaft by a cable-rope (*B*, Fig. 63). The chain hangs in an auxiliary shaft, or a ladderway in the hoist-shaft, and is equal in weight to the full depth of hoist-rope *A*. As the loaded cage begins its

FIG. 63.

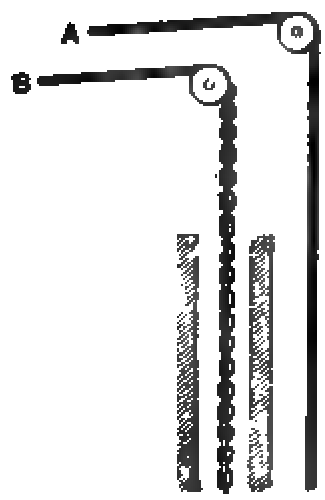
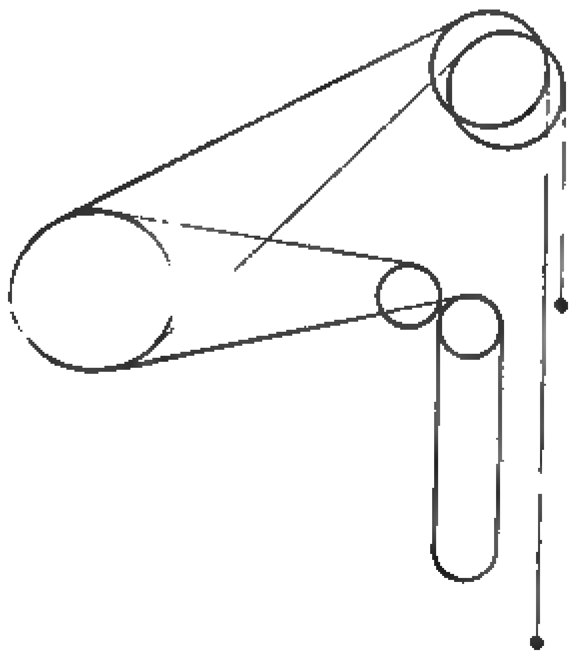


FIG. 64.



Counterbalancing the Rope by Chain.

hoist the entire weight and length of chain is hanging from its drum to assist the engines in balancing the rising load. With the continuance of hoist the cable *B* pays out chain, which coils

on the bottom of the auxiliary shaft. As the two cages approach one another the hoist-ropes nearly balance, the chain is coiling at the bottom, and the cable-rope is being paid out. When the cages are midway, the entire chain is coiled there and all the cable is also paid out. The counterbalancing cable-rope is now wound under the drum and lifts the chain, whose weight now balances the descending rope to the end of its journey. At all points in the hoist the amount of suspended chain balances the differences between the weights of the two pendent hoist-ropes *A*.

In a given case the chain counterbalances a load of 3500 lbs., cage 1300 lbs., car 1540 lbs., in a shaft 3400 feet deep; the radius of the hoist-drum is 6 feet, and that of the chain-drum 27 inches. At the beginning of the hoist 8 tons of chain are suspended from the auxiliary drum. After 400 feet of hoist, 12,134 lbs. are suspended; at 400 feet further, 8400 lbs.; at 2200 feet below the surface, 4666 lbs.; at 1800 feet down, 933 lbs. are yet uncoiled; and at the end of the hoist the full weight of the chain is again suspended. The bore-hole down which the chain hangs is 638 feet deep; 51 H.P. are required to overcome the friction due to this increased weight, but 345 H.P. are saved by balancing the rope. In another shaft, 2200 feet deep, a counterbalance-rope 700 feet long and a chain 580 feet, weighing 4320 lbs., are employed; the size of the chain is graduated to meet the varying weights, and for 104 feet is of $\frac{5}{8}$-inch links, for 162 feet of $\frac{3}{4}$-inch, and for 314 feet of $\frac{7}{8}$-inch.

In Despré's method an endless cable has the balancing chain, and is given one or two turns on the drum between the full and the empty rope (Fig. 64). It passes half-way down an auxiliary shaft. One-quarter the way down the chain is swung on a pivot, and fixed to the cable so that the other end of the chain is at the top of the auxiliary shaft at the beginning and end of the hoist.

The Camphausen System.—A counterpoise (*W*, Fig. 65) is suspended from a supplementary spiral drum mounted on the engine-shaft. The cable-ends are attached to the small and large ends of the spiral. It winds itself on the descending side simul-

taneously with the winding on the ascending side, and thus maintains a balance by the differential lever-arm of its weight. Until the cages meet in the hoistway, the lower branch of the cable travels faster than the upper. During the balance of the hoist the upper winds faster than the lower. For a depth of 1900 feet the auxiliary shaft was 250 feet. The installation is inexpensive and economical.

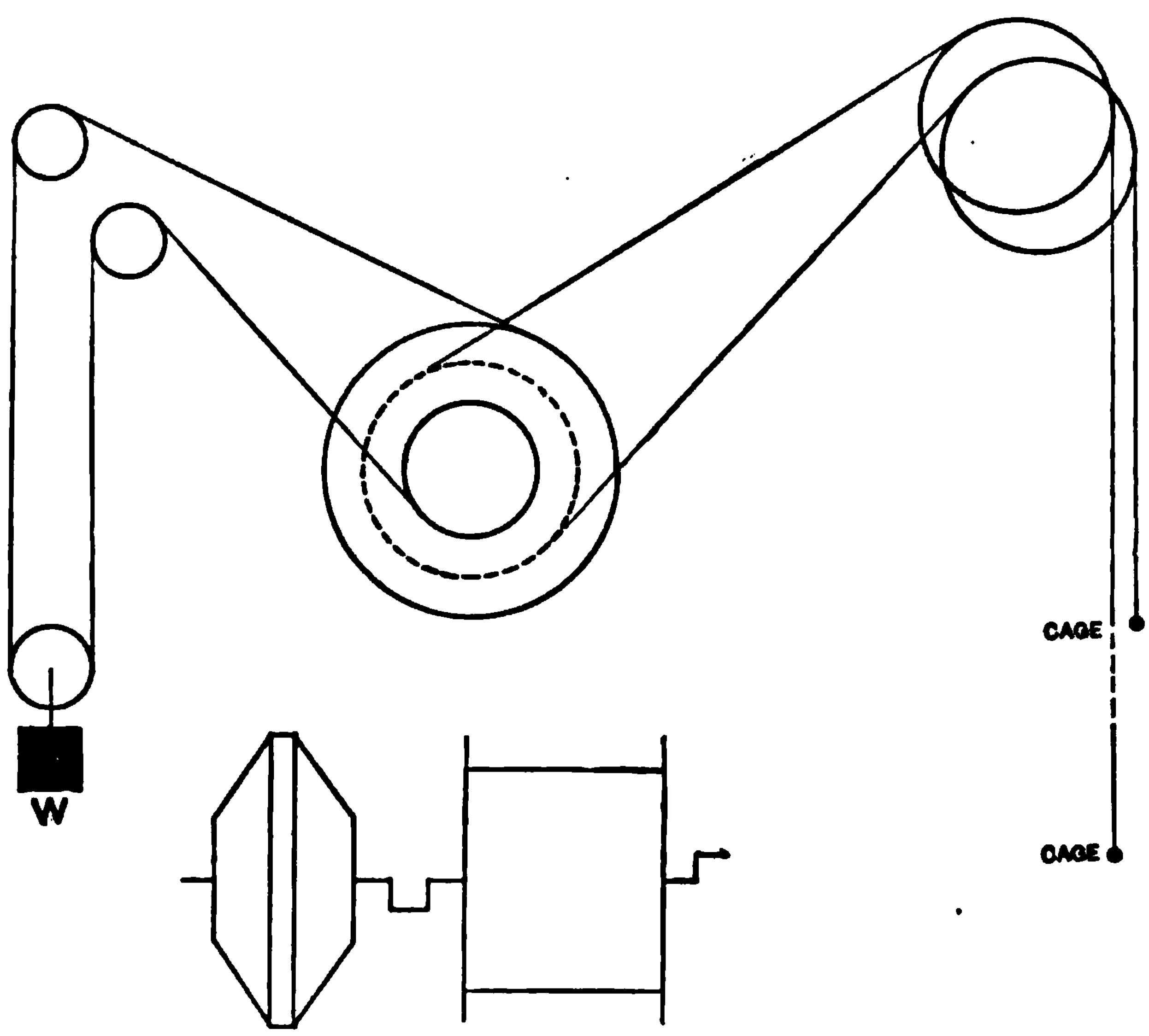


FIG. 65.—The Camphausen System of Counterbalancing.

A very extended discussion of the operations of hoisting is to be found in a monograph of the Institute of Mining and Metallurgy, XI, Part I.

Designing Hoisters.—This step consists in calculating the dimensions of the two cylinders, which are capable (1) of starting the load from the weakest position of their combined crank-pins, and (2) of maintaining an average speed of hoist with it commensurate with the desired output. The mine output is controlled by the capacity of the shaft or the haulage-way, which in turn depends on the size of the car and the number of cars delivered per hour.

Data Necessary in Designing.—From a shaft or a steep slope the cars arrive at the surface singly; from a gentle slope the cars come in trains. The dimensions of the carriers (car, bucket, or skip) and their capacity are known or assumed. The load, q, which a single car can carry is also known, hence the load which falls upon the hoisting- or haulage-engine can be determined. This may be divided into a dead load and a live load, the former being the cage, car, skip, or bucket, while the live load is the mineral and the friction resistances.

The speed depends upon the character of timbering. A mine employing buckets has usually a poorly timbered shaft, in which the speed cannot exceed 300 feet a minute; with skips or slope carriages the timbering must be of a better character, and the maximum rate of 1000 feet is allowed; if cages are employed, the shaft is well timbered and the speed may reach 3000 feet per minute.

The depth of the shaft is known; hence the time required per trip is definitely fixed; and if to it be added the time lost at the top and at the bottom while the buckets, cars, or carriages are being exchanged and loaded, the time occupied in each round trip in each hoistway will be known. The time lost in filling an attached bucket at the bottom or dumping it into the top is from 3 to 5 minutes. If the buckets are detached and promptly replaced by loaded buckets, the time is much less. Skips loaded from chutes are almost as time-consuming as buckets. The skip occupies 2 to 3 minutes to side-track. A car may be run on or taken off the cage or a slope carriage in 25 seconds. These delays seriously reduce the capacity of the shaft and the possible output. Hence for large mines cages and cars are resorted to. The influence of this delay on the aggregate output is diminished by increasing the number of compartments in the shaft or by increasing the velocity. The former is preferable.

Formulæ for Determining the Number and Duration of Trips.—By the following equations may be determined the time occupied per round trip in each hoistway and the number of trips possible per hour in each hoister:

Let z = the number of hoisting-compartments;
D = the depth of the shaft in feet;
v = velocity of the hoist, feet per minute;
v' = velocity of lowering per minute;
Q = output, tons, per hour;
q = load, tons, per trip;
t = the minutes to load and unload at the top or the bottom;
$T = Az$ = minutes per round trip in any hoistway;
A = the interval in minutes between the arrival of each cage, skip, or bucket at the shaft top;
n = number of trips per hour in each hoistway;

$$n = Q \div zq, \quad A = \frac{60}{nz}, \quad \text{and} \quad T = \frac{D}{v} + \frac{D'}{v'} + zt.$$

Thus in a poorly timbered shaft, if only one bucket be run up and down without detaching, the output from a 300-foot shaft is about 3.6 tons per hour. With three buckets in constant use, each holding 600 lbs., the hourly product cannot exceed 6 tons. With excellent timbering double the speed may be permitted, in which event three buckets will deliver 7.2 tons per hour at the surface. So becomes evident the influence which the loss of time at landings has on the output when even doubling the speed increases the former only one-fifth. An additional hoistway were better.

EXAMPLE.—For an output of 360 tons in ten hours a rate of 360 feet is necessary with but one hoistway; with two hoistways and cars holding 3000 lbs. each the rate need be only 144 feet per minute, because of the time saved at the changes. $z=1$; $t=0.41$; $D=300$ feet; $q=1.5$; $Q=36$; $n=24$; $A=250$ min.; $T=2.5$ min.; whence $v=v_1=366$ feet. With $Z=2$, A becomes 2.5 and $n=12$. $T=5$ min.; whence $v=v_1=144$ feet. The horse-power in the two cases would be 100 and 32 respectively. This, at the coal-pile, amounts to 40 tons coal per month.

Determining the Size of Rope.—The rope supports the total load, the friction, and the starting stresses. The latter is larger than is usually suspected. A gentle start gives a stress of 2 per cent only, whereas a start from 12 inches of slack rope would make the stress 1.4 times as great as the actual load. The stress

at starting is usually assumed at 10 per cent of the gross load, including friction. The total tension produced by these resistances must not exceed one-third of the ultimate strength of the rope.

The Bending Stress on the Rope.—The rope must be sufficient for the maximum stress falling on the rope after hoisting begins, including the stress induced by bending the rope over its sheave or drum. This sum is not to be over one-third of the ultimate strength of the rope. Frequently, with a small drum, the bending stress may far exceed that due to the load. The value for the bending stress is represented by the formulæ in Chapter VII.

The Available Strength of a Rope.—After deducting for bending stresses, this should be at least one-eighth of its ultimate strength. The size required may be ascertained from manufacturers' tables.

EXAMPLE.—A 6-strand, 19-wire rope of cast steel 1 inch diameter will be required if it is to support a load of 7270 lbs. over a sheave 44 inches diameter. $k = 14,606$, load and friction $= 8000$, and maximum load is 22,606. Ultimate strength must be 67,800 lbs. Had the drum been 84 inches diameter, k would have been 5583 lbs. and have required an ultimate strength 40,752 for a $\frac{7}{8}$-inch rope.

Determining the Dimensions of Cylinders.—The size of the rope, the dimensions of the drum, and the values for the various loads and resistances to be overcome having been determined, the area and stroke of the piston is made sufficient to start the load with the allowable initial pressure and hoist it at the desired speed. The work of the engine must also be calculated for the average resistances at the mean velocity per minute. The cylinders should not be too large, because the throttling that must ensue for the economic work is wasteful of steam, for the condensation would be excessive. Neither can they be calculated for an overload, as then they might not meet the probable emergency. The cylinders should be apportioned largely according to the conservative judgment of the engineer. Care, however, must be taken to assure a sufficiently large starting moment of the engine to pick up the load. The mechanical efficiency, m, may be used as 0.7

for a slide-valve engine and a cut-off of one-half. The Corliss engine may be calculated to have an efficiency of 0.85 with a cut-off of one-third, for a normal load.

The Hoisting Capacity of an Engine.—The following equations will then be employed for determining the depth to which a given engine can work within the normal limits of boiler pressure which is available and the average r.p.m. for that class of engine. They will also serve to calculate the required dimensions of one or two cylinders for given conditions:

L = the length of the drum, inches;
r = the smaller diameter of the drum, inches;
d = the larger diameter of the drum, inches;
n = the number of grooves in the drum;
X = the diameter of the rope, inches;
R = the weight of the rope in pounds per foot.
v = the velocity of hoist, feet per minute, $\frac{6.283Nr}{g}$;
A = the weight of bucket, cage, car, skip, or carriage lbs.;
M = unbalanced load on the drum at the start;
K = the moment of resistance at starting;
W = the average load during the hoist;
F = the friction of the gross load, including the acceleration of the moving parts;
g = gear ratio, second-motion engine, $= \frac{y}{x}$;
y = number of teeth on the gear-wheel;
x = number of teeth on engine-shaft pinion.

The capacity of the engine may be known from its indicator cards or from a knowledge of the cylinder dimensions and the proposed boiler pressure, P, and engine speed. The velocity for a given output, number of trips, and depth will have been determined. Then

$$H = 0\ 000003966csk^2mpN = Wv.$$

Or, the value for k and s may be ascertained for a given set of conditions of load and speed as well as for P and N. The ratio of s to k is assumed between 1.3 and 2, according to the class of engine desired.

With a direct-acting engine $g=1$. In all cases the mean piston speed is $0.31841\frac{sVg}{d}$.

Hoisting on Inclines.—If the hoist is not in a vertical shaft, the load, M, must be multiplied by the sine of the angle of the slope (the percentage of inclination), and the value for F is multiplied by the cosine of the slope angle, in calculations for the cylinder dimensions.

The coefficient of friction, f, is 4 per cent with a bucket in a vertical shaft, 6 per cent with a cage or skip, and 8 per cent with a bucket in a slope. All of these are reduced to the pull at the circumference of the drum.

The Starting Moment of an Engine.—In designing an engine to start the unbalanced load, M, it is necessary to determine by the equilibrium of moments the value for the moment of the load, friction, and starting stress, and equate this with the minimum rotary effort upon the crank-pin. The starting moment of the engine is ascertained by the substitution, in the following formula, of the value for c, corresponding to the given cut-off and ratio of connecting-rod to crank-arm.

The minimum rotary effort, R, of a duplex single-expansion engine $=CsP_1k^2mg=R$, inch-pounds. The maximum rotary effort $=R'=C_1'sP_1k^2mg$. s and k are in inches, P_1 is the net initial pressure, $P-B$, and C, the coefficient representing the minimum rotary effort as ascertained from the following table for the particular ratio of connecting-rod, l, to crank-arm, a, and a piston clearance of 7 per cent.

The values given are the rotary efforts exerted upon the two crank-pins for each pound of net initial pressure for each inch of one piston diameter, the pair having their cranks at quarters. The clearance is assumed to be 7 per cent, and a back pressure of 18 lbs. absolute is allowed. If the engine is a con-

densing engine, with 3 lbs. absolute back pressure, the values for C' and C are somewhat larger than those in the table. A pair of cylinders of 10 inches diameter and 20 inches stroke with connecting-rods 50 inches long, receiving an initial steam pressure of 100 lbs. absolute, will have a minimum rotary effort equal to 48,480 in.-lbs., if the cut-off be assumed at $\frac{1}{3}$ and the back pressure be 3 lbs.

TABLE OF MAXIMUM AND MINIMUM ROTARY EFFORTS FOR CERTAIN RATIOS OF CONNECTING-ROD TO CRANK-ARM. DUPLEX ENGINES.

Values for C_1 and C.

Apparent Cut-off.	$l=4.5a$.		$l=5.5a$.		$l=7a$.	
0.00	0.5615	0.3974	0.5600	0.3974	0.5580	0.3974
$\frac{1}{2}$	0.4664	0.3633	0.4606	0.3688	0.4586	0.3696
$\frac{1}{3}$	0.4380	0.2472	0.4365	0.2499	0.4350	0.2550
$\frac{1}{4}$	0.4256	0.1878	0.4193	0.1923	0.4124	0.1966
$\frac{1}{5}$	0.3893	0.1469	0.3818	0.1473	0.3786	0.1512

The hoisting-engine must be able to start the load from any point in the shaft and at any point in its stroke, and hence should have in its weakest position a rotary-effort moment greater than that of the maximum load which may fall upon it. Doubtless all the masses involved will have attained sufficient velocity in one revolution to produce an average effort on the applied moments from both cylinder-cranks. The engine-friction moment decreases during the hoist nearly commensurate with the load moment.

Hoisters cannot be regulated by fly-wheels, and their sole means of maintaining a uniform rotary effort is that which is afforded by the drum or a weight of the connecting-rod. If these are made heavy, the velocity may be somewhat uniform. But inasmuch as the engine must furnish an excess of power at the beginning which is 50 per cent more than the average, and the inertia of the rope, sheaves, drum, etc., must be overcome, and, later, the kinetic energy absorbed by the brake, some

form of energy accumulator should be devised for the purpose of economizing power.

Determining the Size of the Drum.—The minimum allowable diameter of the cylindrical drum is fixed by the size of the rope (Chapter VII), it must be more than forty times the diameter of the rope—one hundred times the diameter would be preferable, since thereby it would reduce the bending stress of the rope and increase the net working load. The smaller end of the cone or fusee, and of the barrel of the reel, is also fixed by the same ratio. The desired hoisting speed also influences the final decision as to the diameter. The drum or drums are geared to the same shaft and placed centrally between the cylinders.

The length of the drum is controlled by the depth of the hoist and its location with reference to the shaft. On cylindrical drums the rope coils are contiguous to one another. Conical drums are provided with spiral grooves. Reels have the successive coils of rope winding on one another. The length of the drum and its distance from the shaft must be such that the acute angle from the sheave to the ends of the drum must not exceed 6°. When this fleet angle exceeds the limit, the rope mounts its adjoining coil, and provision must be made for guiding the rope upon sheave and drum, or the drum must be set back from the shaft. In the first case the engine may be set on a carriage to move with a varying position of the rope on the drum.

The cases are rare where but a single rope with a single cylindrical drum is sufficient. Two ropeways are the rule with two independent cylindrical drums; or a double cylindrical drum, two conical drums, or a double cone, the fusee, or two reels are employed according to the requirements in the given case. These may be operated independently of one another, though provision is always made for locking the pair of drums together to obtain simultaneous hoist and lowering. This is true whether the engine be a direct-acting or a second-motion engine.

The General Formulæ for Hoisting Calculations.—The following table then gives the value for the starting resistance of the

load for any variety of drum, the moment of which is equated with the starting effort of the engine, to determine the dimensions of the cylinders. The value for the unbalanced load to be hoisted throughout the trip at a requisite velocity is also given for the selected drum. Then the resistance $M+F$, multiplied by the velocity v, is equated with the value for engine power, to determine the engine dimensions for the dynamic load.

THE STARTING MOMENTS, WORKING LOADS, ETC., WITH VARIOUS HOISTING ENGINES AND DRUMS.

	Maximum Unbalanced Load, M.	Friction of Load, F.	Minimum Admissible Radius of Drum d, Inches.
One-cylinder drum. .	$A+RD+2000q$	Mf	$25X: \frac{1909Dg}{N}$
Two- " " . . .	$RD+2000q$	$(M+2A)f$	"
Double conical drum.	"	"	$r+p(n-1)\frac{(M+2A)r}{2A+2000q}$
Fusee.	$2000q$	$(M+2A+RD)f$	$\frac{RD+A}{A}r$
Reel.	"	"	$r+nt; \sqrt{3.82Dt+r^2}$
Koepe system.	"	$(M+2A+2RD)f$	$25X$

	Number of Grooves, n.	Length of Drum, L, Inches.	Starting Moments, K, Inch-lbs.
One-cylinder drum. . .	$\frac{1.9098D}{d}$:	nX	$(M+F)d$
Two- " " . . .	$\frac{3.8196D}{d}$:	$2nX$	$(M+F)d$
Double conical drum.	$\frac{3.8196D}{r+d}$	$P(n-1)$	$(M+A)(1+f)r - A(1-f)d$
Fusee.	$\frac{\sqrt{\left(\frac{M}{R}\right)^2 - r^4}}{12.566r^2}$ $\frac{\sqrt{\left(\frac{M}{R}\right)^2 - d^4}}{12.566d^2}$	$(n+1)X+\frac{1}{2}n$	$(M+A+RD)(1+f)r - A(1-f)d$
Reel.	$\frac{d-r}{t}$; $\frac{3.82D}{d+r}$	. . . .	"
Koepe system	. . . .	. . . .	$(M+F)d$

The following examples will suffice to illustrate the use of the formulæ and tables for calculating the work of hoisting.

1. *Duplex engine, single drum.*

In a shaft 400 feet deep an output of 100 tons per ten hours is desired from a single compartment. The load in a car is $\frac{3}{4}$ of a ton; weight of car and cage, 1800 lbs. Assuming a duplex engine, single drum, and full steam pressure available for starting, required the size of the cylinders, number of trips, etc.

A 19-wire rope at starting (Chapter VII), on an 8-foot sheave, suffers a stress $K=A+2000q+F+RD$. Assume the rope to weigh 1 lb. per foot. The total load $(1800+1500+400)(1.1)=$ 4070 lbs. This requires a $\frac{3}{4}$-inch cast-steel rope weighing 0.9 lb. per foot, because the total load must be less than $\frac{1}{7}$ the ultimate strength, which must be 28,490 lbs.

$$k=\frac{367000x^3}{r+0.5x}=3200.$$

3200 lbs. is less than one-third the ultimate strength.

Output per hour, 10 tons; $q=\frac{3}{4}$; hence, with single cage, $n=14$ trips; time, 4.3 min. for round trip. Allowing for loss at top and at bottom of 30 seconds each, the hoisting time is 1.65 each way and $v=360$ feet per minute. A drum of 4 feet diameter makes 28.6 r.p.m. Let $f=0.1$, $m=0.75$, the engine stroke $=1.5k$, and the mean cut-off $=\frac{1}{2}$.

$$(1800+400+1500)(1+0.1)360=44.4 \text{ H.P.};$$
$$0.00000c3966sk^2Nmp\times 2=44.4;$$

whence $k^2p=174{,}012$ and $k=13.5$ inches.

With 7 per cent clearance $p=0.8925(100)-18=71.25$ lbs. per square inch. Cylinder is $17\frac{1}{4}'' \times 26''$, nominally, with counter-bore at each end of $1\frac{7}{8}$ inches.

For starting, the dimensions should be (page 10)

$$CsP_1k^2mg=R=97{,}710 \text{ inch-pounds};$$

$C=0.3688$; $g=1$; $P_1=82$; whence $k^2s=4308$ and $k=14.25$ inches.

2. *Double drum and duplex engine.*

A shaft is 2000 feet deep; car and cage, 5000 lbs.; load, $2\frac{1}{2}$ tons; velocity of hoist, 2000 feet; gear ratio, 1; drum, 60 inches radius; engine-stroke, 48 inches; modulus, 0.8; admission steam pressure, 100 lbs. gauge; back pressure, 5 lbs.; cut-off average, $\frac{1}{2}$; sheave, 12 feet; and f, 0.1. Required the size of 19-wire rope, diameter of engine cylinder, and output of mine.

At the same velocity of lowering as of hoist, the round trip is made in three minutes, including stops. In each hoistway 20 trips are made hourly. The output is 1000 tons daily.

The starting stress is $(5000+5000+RD)(1.1)+K$; assuming a weight of 7-wire rope of 3 lbs. per foot, and a diameter of $1\frac{3}{8}$ inches, the load is 17,600 lbs., and the bending stress 13,060 lbs. The aggregate starting stress must be less than one-third the ultimate strength. That for a $1\frac{3}{8}$-inch cast-steel rope is 62 tons.

Dimensions of cylinders for normal running:

$$F=(6000+15{,}000)0.1=2100 \text{ lbs.};$$

$$(6000+5000+2100)2000=26{,}000{,}000 \text{ ft.-lbs.}=793.6 \text{ H.P.};$$

$$2\times 0.0000003966sk^2Nmp=793.6.$$

The depth of shaft equals in this case the velocity of hoist, hence the number of coils = the r.p.m.

$N=63.6$ r.p.m.; piston speed $=508.8$; whence $k^2p=40{,}966$.

For an average cut-off $\frac{1}{2}$ with 3 per cent clearance,

$$p=0.8658(100+14.7)-5=99.3;$$

$k^2=397$ and diameter of cylinder $=23$ inches.

At 63.6 r.p.m. for $v=2000$ feet, the diameter of drum is 9.91 feet. The starting resistance is $17{,}600\times 59.5=0.3688\times 48\times 109.7\times 0.8\times 1\times k^2$; whence $k=26$ inches.

With these dimensions of cylinders the M.E.P. at the average rate may be 56 lbs., corresponding to a cut-off of 0.18 for 3 per cent clearance:

$$56=C(114.7)-5, \text{ whence } C=0.532.$$

Or, at a cut-off of $\frac{1}{2}$, an initial pressure of 71 lbs. absolute will suffice. $56=0.8658(P)-5$; $P=71$.

The normal rate of running being at $\frac{1}{2}$ cut-off, the Corliss engine of $26'' \times 48''$, giving 733.3 H.P., will consume 17,600 lbs. steam per hour.

3. *Double conical drum and duplex engine.*

Same specification as in 2.

The smallest diameter of the drum is assumed at 80 inches, for the rope of $1\frac{3}{8}$ inches. The angle of the cone is $14° 30'$; $p=0.47$; $P=1.825$; $c=230.6$ inches; $p'=4.178$.

To maintain equal moments of resistance when the load is at top and at bottom,

$$d=\frac{5000+2\times6000+2\times5000}{2\times5000+5000}40=72 \text{ inches.}$$

$n=\dfrac{3.8196\times2000}{40+72}=68.2$ grooves; $L=1.825\times68.2=10$ ft. 4.5 in. This is rather a long drum. To obtain a fleet angle of 6° it must be 100 feet from the shaft.

The horse-power required for the normal rate of running is, as before, 763.9; $N=68.2$ r.p.m.; piston speed, 545.6 feet; and $p=99.3$ lbs. per square inch; k becomes 19.5 inches.

The starting resistance, neglecting the lengths of rope above the surface landing to the head sheave, requires a diameter, k, of 15.6 inches

$$(5000+6000+5000)(1.1)40-5000(0.9)72 = .3688\times48\times109.7\times0.8\times k^2.$$

4. *Reels with duplex engine.*

Same specifications as in 2.

Barrel diameter is 40 inches and flat rope is $\frac{1}{2}\times5\frac{1}{2}$, weighing 4.2 lbs. The bending stress is

$$k=\frac{89465t}{R+2.23t}=2118 \text{ lbs.};$$

the diameter of the outer length of rope is

$$d=\sqrt{3.82Dt+r^2}=65 \text{ inches};$$

and

$$n=2(65-20)=90 \text{ coils.}$$

If the cylinders are direct-connected, the piston speed will be 720 feet per minute, whence $k=17.08$ inches. The above speed is rather high. If the barrel be enlarged to 60 inches, then d becomes 97.5 inches, $N=60$ r.p.m., and $k=20\frac{3}{4}$ inches.

The starting moment of the engine at an assumed cut-off of $\frac{1}{2}$ would be

$$R=0.3688\times48\times82\times571.8\times0.8\times2=1,328,000 \text{ inch-pounds.}$$

The moment of the starting resistance is

$$(16,000)(1.1)20-(5000)(0.9)65=94,700 \text{ inch-pounds.}$$

5. *Koepe system with duplex engine.*

Same specifications as in 2.

The load upon the engine is at all times 5000+ the friction of the gross weight, moving at 2000 feet per minute. Neglecting the weight of the 12-foot sheaves, and assuming the rather large frictional coefficient of 0.1,

$$F=(5000+6000+10,000)(0.1)=2100 \text{ lbs.};$$

$$\frac{(5000+2100)2000}{33,000}=430.3 \text{ H.P.};$$

$$430.3=2\times0.000003966sk^2Nmp.$$

$N=53.05$ r.p.m., if $g=1$; and the piston speed $=424.4$ feet, whence k becomes 17.05 inches.

If the drum is geared with a ratio of 2, k would become 14 inches. Then $(5000+2700)72=0.3688pk^2smg$, the minimum rotary effort of the engine with a cut-off of $\frac{1}{2}$ being 0.3688.

Electric Hoisters may be located in relation to incline or shaft exactly where they can be operated to the greatest advantage, considering convenience of position, without the pipes inseparable from air- or steam-hoisters. Their speed is constant, the motor is economical, and the only power consumed is that actually required to handle the load. One objection to their use is the large gear ratio which is required to bring the

speed of motor down to speed of hoister (Fig. 66). But they have none of the ills of the engine, with its tremendous vibration, rattle, heat, and noise of exhaust, or of the compressed-air hoist, which has, in addition, the extremely cold exhaust.

FIG. 66.—An Electric Hoist.

The motors for electric hoist are either of the continuous-current, series- or compound-wound types, or of the alternating-current induction type, designed for either constant or variable speed, and geared to the drum. They are fitted with appropriate rheostats. The direct-current series motor is particularly adapted to cases which demand a very high starting torque and where closeness of speed regulation is not imperative. The compound-wound motor should be employed when there is liability of the motor racing on throwing off the load. It has a high starting torque, and its speed will not rise above a certain definite value when the mechanical load is thrown off.

The variable-speed induction motor is best used for hoisting. Its starting torque is quite high and its speed regulation sufficiently flexible to make it very desirable.

Calculating the Size of Wire for a Hoister.—One of the three variable quantities are to be determined: the horse-power of the motor, the load, M, and the speed of hoist, v.

The mechanical efficiency, m, for the direct-current motors is 0.85 and may be regarded as unity for the induction motors.

The effective horse-power of the motor must equal

$$0.0000303v(M+F)$$

and

$$M+F=33000\times\text{horse-power}\times\frac{m}{v}.$$

In determining the size of wire for the direct-current circuit the formulæ given in the next chapter will be used.

EXAMPLE.—It is desired to operate a 30-horse-power moto at a distance of 500 feet from the generator by a direct current of 440 volts. Then $D=500$; C, for 1-horse-power (Chapter VI), is 2 amperes for 440-volt circuit. Let V, the loss in volts, be 10 per cent, or 44 volts. Then, for a 30-horse-power motor, 30×2 amperes (=60 amperes) will be required. Substituting in the formula, the wire must have 14,300 circular mils and, according to the table, will be of No. 8 size. The weight of the wire will be 69 lbs. in the total, and its cost $17.25.

If the current be a three-phase alternating, the value for K being assumed as 17, a No 10 wire with 11,600 circular mils will be sufficient. Its total length of 1500 feet, weighing 73 lbs., will cost $18.25.

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CHAPTER VI.

ELECTRIC GENERATION AND WATER-POWER.

The Application of Electricity to Mining. — Though the utility and economy of electricity for underground work have long been recognized by the mining engineer, the conservatism forced upon him by restrictive legislation has made its installation underground one of slow growth. Its economy and convenience have been recognized; its adaptability to all operations of mining, lighting, blasting, coal-cutting, hoisting, haulage, etc., its flexibility to all distances, is conceded; it never vitiates the air, as do steam-engines, nor does it fog or chill it, as does compressed air; and it is capable of automatic regulation, requiring but a small number of employees. The low cost of maintenance and repairs, its high economy, as compared with other motors, and the fact that there is no leakage of power when the motor is not in use, gives positive assurance of future benefit, especially where the power is to be intermittently applied. It is capable of transmission over great distances with small loss, and to localities where the haulage of coal for steam generation is exceedingly difficult. In this field it is unrivalled. The development of the steam-turbine and impulse-wheels makes possible direct connections with the electric generator, forming an ideal combination in point of economy of power and space.

Two serious objections obtain against electric installations for mining purposes, and they are all-important. The first is the risk of fire and shock of those coming in contact with the electric current. They cannot be wholly eliminated without very great cost of copper and its insulation. Nevertheless it is possible to

insulate to a high degree, to place the wires remote from accidental contact by the men, and to reduce the fire risk by the use of non-sparking devices and enclosed machinery. The partial success in this direction has removed the chief objections to underground installations. A less potent objection is that, electricity not being applicable for every operation of mining, the owner is compelled to maintain two separate power systems where percussion-drills are needed and large bodies of water are to be pumped.

Electrical Units.—The electric units are the *ampere* and the *volt*, corresponding to quantity and pressure. The unit of resistance, R, to the electric flow is the *ohm*, whose value corresponds to a unit electromotive force and permits the flow of a unit of current. The standard ohm is the resistance offered by a column of pure mercury at 0° C., of uniform cross-section, 106.3 cms. long and 14.4521 grs. weight.

Electromotive Force.—This is the electric pressure which forces a current through a resistance, or is the difference of potential between its termini. The unit of *electromotive force*, E.M.F., is that pressure which will force a unit current through a unit resistance. The unit is the *volt*, which is 0.6974 of the difference of pressure between the poles of a Clarke cell at 15° C., whose E.M.F. is 1.434 volts.

Current Capacity.—An electric current, I, has unit strength when a resistance of one ohm will afford an E.M.F. of one volt between its ends. The unit is the *ampere*, that current which will electrolytically deposit silver at the rate of 0.0011118 gr. per second.

The quantity of electricity which passes through a given cross-section of an individual circuit in t seconds when a current of I amperes is flowing is equal to It units. The unit is the *coulomb*, whose value equals the flow of one ampere for one second. One ampere for one hour equals 3600 coulombs.

Capacity, C, is the property of a material for holding a charge of electricity. A condenser has unit capacity when a constant of electricity will charge it to a potential of one volt. The *farad*

is the name of this unit, but since its value is much greater than actual practical values, its millionth part, the micro-farad, is used as the practical unit.

Electric Energy, W, represents the work done in a circuit or conductor by a current flowing through it. The *joule* is the name of this unit, its value being the work done by the passage of one ampere through one ohm for one second. Electric power, P, is one joule per second and represents the passage of one ampere of current under a pressure of one volt. Its value is measured in watts. One *watt* per second is a joule.

$$1 \text{ watt} = 0.7373 \text{ ft.-lbs.}$$
$$746 \text{ watts} = 1 \text{ horse-power.}$$

In order to avoid the use of large numbers, the term *kilowatt*, or 1000 watts, is universally used in power and lighting. Allowing for 15 per cent drop, the following table has been computed, showing the quantity of current necessary to deliver one horse-power at the various stated pressures:

TO DEVELOP ONE MECHANICAL HORSE-POWER AT THE MOTOR SHAFT.

8 amperes	110-volt circuit.
7 "	125 " "
4 "	220 " "
3.5 "	250 " "
2 "	440 " "
1.85 "	475 " "
1.75 "	500 " "
1.6 "	550 " "

Ohm's Law.—Resistivity, ρ, is the specific resistance of a substance, and is the resistance, in ohms, of a cubic centimeter of material to a flow of current between opposite faces. In any electrical circuit, the current which flows equals the electromotive force divided by the resistance. Expressed in the form of an equation, Ohm's law is $IR = E$, in which I is the number of amperes flowing in an undivided circuit, E the algebraic sum of all the E.M.F. in the circuit, and R the sum of all the resistances in series in the circuit.

Since conductors offer more or less resistance to the passage

of current, there will be a drop or fall of potential, E_d, along the circuit, the amount of which will depend upon the resistance of the conductor and the current that is flowing:

$$E_d = IR.$$

The resistance of a conductor may be found from the equation

$$R = \frac{\rho L}{A},$$

in which ρ is a constant, termed the *resistivity*, whose value depends upon the material and the temperature of the conductor; L is the length in centimeters, and A the cross-section in square centimeters. The resistance of pure metals increases with a rise in temperature. A rise of 1° C. increases the resistance by 0.004 times that at 0° C.

Circular Mils.—A circular unit is a circle 0.001 inch in diameter. A conductor one foot long and one circular unit cross-section is called a mil-foot.

Electric Currents.—Two forms of currents are generated and transmitted for power purposes—the continuous current and the alternating current. In the former, known also as the direct current, the electric fluid maintains a flow in one direction through the circuit. An alternating current is one which changes the direction of its flow at regular recurring intervals. Dynamos generating electrical energy are of two classes, according to the currents to be produced.

Electric Generator.—The continuous-current dynamo consists of field-magnets, an armature, and a commutator. The field-magnets are poles of iron arranged in pairs. The dynamo may have two field-magnets, when it is known as the bipolar machine, or several pairs of field-magnets, then called a multipolar machine. In the first case the armature revolves between the opposite ends of two magnets, the other ends being directly connected. In the multipolar machine the magnets project inward around a ring, or casing, inside of which, and nearly touching, the armature revolves. Each pole is wound with wire, through which a

current flows to maintain the magnetism in the poles. The number of poles may be 2, 4, 6, 8, or more, the wire being wound right-handed about the first pole and left-handed about the second in order to produce with the same current the alternate polarity in the respective pieces.

Fig 67 —A Drum Armature.

The armature may be of the drum or the ring type, the former being stronger and generally employed for power transmission. A rotary shaft (Fig. 67) is built to a solid core of thin plates of soft iron projecting radially and insulated from one another by mica or paper. They are bolted together with the surface slotted longitudinally. In the slots are laid coils of wire whose projecting ends are connected with copper strips composing the commutator. Fig. 68 illustrates the ring type of armature.

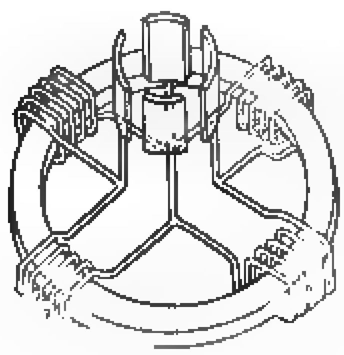

Fig. 68.—Ring Armature.

The commutator has an equal number of copper insulated strips, held firmly in place by a bronze sleeve. Brushes collect the current from the commutator, one set for each pair of poles. They are composed of wire, or of small blocks of carbon coated with copper.

The Winding of Generators.—According to the method of winding the coils, there are series-wound machines, shunt-wound machines, and compound-wound machines.

In the series-winding, the arrangement of coils provides for the whole current flowing continuously and directly from one brush through the winding of the magnets, then through the external circuit and back to the other brush. The wire around the field-magnets is of large diameter and low resistance. This form of winding is suitable only where a constant current is required, as for arc lighting (Fig. 69).

In the shunt-winding, two paths are open to the current. Leaving one brush, one branch flows through the external circuit and the other through the field-magnets. Both join at the other brush before returning to the armature. This also gives constant E.M.F., but with varying conditions in the external circuit. The fields are wound with a large number of turns of small wire (Fig. 70).

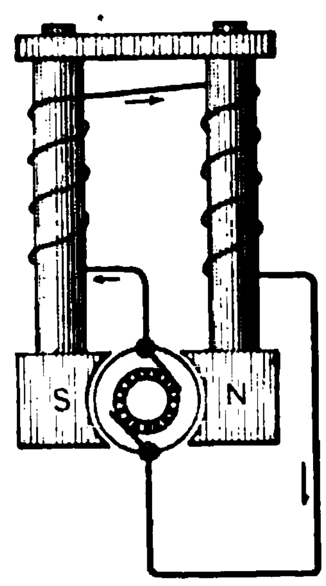

FIG. 69.—Series Dynamo.

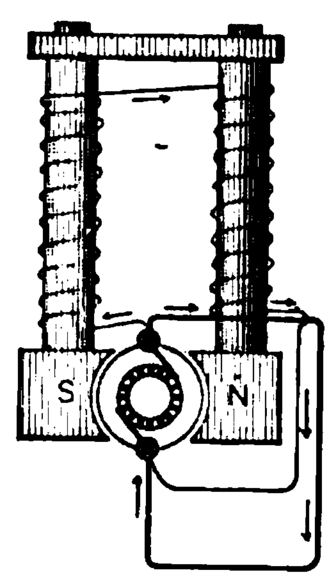

FIG. 70.—Shunt Dynamo.

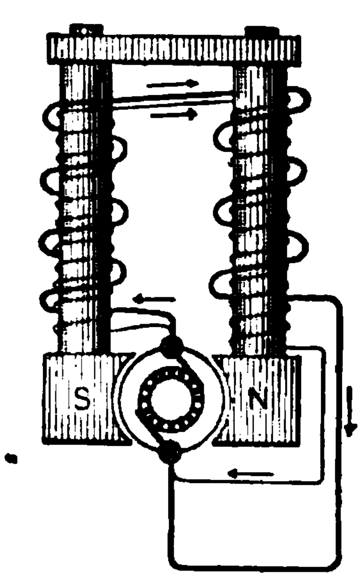

FIG. 71.—Compound Dynamo.

The compound-winding is a combination of both the shunt and the series. The dynamo gives a constant E.M.F. The shunt coils are of numerous turns of thin wire overlaid by a few turns of series coils of thick wire. Both are coupled up (Fig. 71).

The Theory of the Dynamo.—The dynamos are driven by belts from the engine, or may be directly coupled to an engine, water-wheel, or steam-turbine. Belt connection is cheaper than the direct connection.

The rotation of the armature presents successively the plates

of soft wire in apposition to the poles of the field-magnets. This develops in each coil about the corresponding plate a small current of electricity, which is conveyed to their ends at the commutator. The aggregation of the currents thus successively produced in the multitude of coils by a high rotary speed is a current of high potential and large quantity, which may be drawn by the brushes to the external wire circuit. The E.M.F. developed is proportional to the number of turns of wire in the rotating armature.

Two wires from the brushes to the electric motor or lamps complete the external circuit for the current flow.

Alternating-current Machines.—The wiring of the alternating machine is similar to that of the direct-current, but the commutator is replaced by two insulated rings on the armature shaft, one end of the numerous armature coils being attached to one ring and the other end to the other ring. The field-magnets on a bronze base around the rotor constitute the stator. Their north and south poles alternate, their windings being excited by a similar continuous-current machine driven from the alternator.

The armature of the alternating machine is called the rotor. It carries the secondary circuit and has as many coils systematically arranged around the laminated soft steel discs as there are pairs of field-magnets. In its revolution an electric current is generated in the armature coils, which changes its direction as often, during each revolution, as there are pairs of field-magnets. This gives rise to the alternating currents.

If a second set of coils, similar to the first, but independent of them, be wound in the spaces between the first set on the armature, then, by the revolution of the rotor, are induced two distinct sets of currents, one leading the other by one-half a period. Such a current is known as a two-phase current. If three sets of independent coils be arranged equally around the armature, then a three-phase current will be obtained, with its pulsations having their maximum one-third of a period from one another. The three-phase current is sometimes called a polyphase or multiphase current.

The Frequency.—The value of the current varies in the different intervals from zero to a maximum, diminishes with a like regularity to zero, and rises to the same maximum in the opposite direction, returning finally to zero. One such complete interval is a cycle, the tilde (~) being used to denote it. The time in which the cycle of changes of such currents is completed is known as its period. The number of cycles completed per second is its frequency. The frequency usually adopted in practice for power transmission is usually between 25 and 40 cycles. The number of cycles completed in a minute is known as its rate of alternations.

As the points of maximum pressure are removed 120° from that of the other circuits, the algebraic sum of the phase currents, when balanced, is at every instant equal to zero. Two wires should be employed for each of the currents and three return and three direct wires for the three-phase system. But, for the reason stated above, the three return wires may be dispensed with. Three wires then serve for the practical three-phase circuit.

The Double-current Generator.—This is wound to produce either or both varieties of current, and is used where the cost of two separate dynamos has deterred many from introducing electric power. There is a definite ratio between the voltages on the direct and alternating sides. For the two-phase it is 0.76 of that of the direct current which it could produce, and 0.65 of the voltage if a three-phase current. Hence the double-current dynamo is a low-voltage machine.

The difference of potential which can be obtained from a D.C. generator cannot exceed 1000 volts, because of the inability to prevent short-circuiting at the commutator. The alternating-current machine is not limited in voltage from this source of leakage.

A serious defect of the D.C. machines is the sparking of their brushes, due to faulty alignment, an insufficient bearing or imperfect contact, and the short-circuiting in the windings. These faults suggest simple remedies without further mention.

The heating of the coils occurs when any current is forced through the coils, but no harmful results ensue unless the temperature rises above 80°. When exceeded, the insulation softens and is burned. This may be detected by the odor of burning shellac or rubber. Dampness may produce the same results.

The copper bars of the induction machine are all short-circuited, rendering attention to insulation unimportant.

Properties of Copper Wire.—The manufacturers' tables furnish the data for wires of various diameters by which may be known their resistances to the passage of a current. Below is a brief table which will aid in the solution of problems connected with the wiring of a mine for motors.

Wire No. B. & S.; American and Birmingham Gauges.	Circular Mils.		Weight per 1000 Feet of Common Insulated Wire, Pounds.		Safe Current-carrying Capacity in Shaft or Tunnel Work, Amperes.		Resistances per 1000 Feet in Ohms. at 60° F.	
	B. & S.	B.	B. & S.	B.	B. & S.	B.	B. & S.	B.
000	167,806	180,625	603	647	295	315	.0606	.0564
00	133,076	144,400	483	517	250	272	.0764	.0755
0	105,534	115,600	385	415	210	227	.0964	.0881
1	83,692	90,000	308	325	180	193	.1219	.1131
2	66,371	80,686	246	288	150	161	.1529	.1262
4	41,741	56,644	152	198	110	118	.2446	.1797
6	26,251	41,209	102	148	80	85	.3879	.2471
8	16,509	27,225	69	101	60	64	.6214	.3769
10	10,383	17,956	49	71	40	43	.9785	.5670
12	6,529	11,881	35	51	30	....	1.5520	.8569

The Electric Symbols as applicable to motors are:

E = volts at the terminals;

V = volts loss in transmission;

$E+V$ = E.M.F. at the generator terminals;

I = current required at the motor to deliver H, mechanical horse-power at the shaft of the motor;

D = single distance between motor and generator in feet;

L = aggregate length of conductor in feet;

H = the number of mechanical H.P. delivered at the motor shaft;

A = the area of cross-section of conductor in circular mils;

d = the diameter of the wire in circular mils;

R = the total conductor resistance in ohms for the length, L, or $2D$;

W = the weight in pounds of copper conductor;

m = the commercial efficiency of motor in per cent;

" " " " generator, 90 per cent;

S = " " " " whole circuit, per cent;

p = per cent of energy lost in system.

Continuous-current Distribution.—The following rules are serviceable as guides in calculating the dimensions of D.C. conductors. The E.M.F. varies directly with the amount of energy transmitted.

Knowing the work to be done, line loss and E.M.F. at the terminals and point of distribution, then the cross-section of the conductor varies directly with the distance, and the weight, as the square of the distance.

The conditions remaining as given, the weight of conductor will vary inversely as the square of the E.M.F. at the terminals. Given the amount of power which is to be transmitted by a stated weight of conductor and loss in distribution, then the distance over which the power can be transmitted varies directly as the E.M.F. For a given amount of power and a given conductor it varies as the square of the E.M.F.

The Resistance of Conductors.—The resistance of a mil-foot of pure copper at 0° C. = 9.59 ohms.

The resistance of a mil-foot of 96 per cent conducting *Cu* (which is the commercial conductivity usually specified at 70° F.) = 10.81 ohms.

Then the electrical horse-power at the motor terminals $= \dfrac{H}{m}$.

$$H = \frac{IE}{746} \quad \text{and} \quad I = \frac{746H}{Em} \text{ amperes.}$$

The resistance of the conductors, both ways, is

$$R = \frac{21.62D}{A} = \frac{10.80L}{d^2}.$$

The line-drop

$$V=IR=\frac{21.62ID}{A} \quad \text{and} \quad A=\frac{16128.5HD}{Emv}.$$

EXAMPLE.—A motor of 50 horse-power of 0.93 efficiency is to be installed in a mill with the terminal volts at 500. The transmission distance is 500 feet, and the allowable loss in that line 6 per cent. Required the size of conductors. The E.M.F. at dynamo$=\frac{500}{0.94}=531.9$ volts.

The line-drop $V=531.9-500=31.9$ volts, and the area of the conductor

$$=\frac{16,128.5\times50\times500}{500\times0.93\times31.9}=27,863 \text{ circular mils.}$$

The current $I=\frac{746\times50}{500\times93}=80.6$ amperes.

The National Code allows only 97 amperes for No. 5 B. & S. gauge conductor. To transmit 80.6 amperes safely, at least No. 4 wire would be required. For underground work a No. 6 wire would be permissible.

The resistance of No. 4 B. & S. wire $=0.2480$ ohms at 20° C.

The resistance of 500 feet of No. 4 wire $=0.1240$ ohms at 20° C.

Then the volts-drop $=IR=80.6\times0.1240=99.94$ volts; hence volts at dynamo $=500+99.94=599.94$ volts; and per-cent drop $=\frac{99.9}{599.9}=1.870$ per cent.

General Wiring Formula for Alternating-current Distribution.—

Let D = distance of transmission one way in feet;

W = total watts delivered to consumer;

p = per cent of power, W, lost in line;

E = voltage between main conductor at the consumer's end of circuit;

K = constant, whose value depends on the kind of system and the power factor table, page 178;

A = factor for determining weight of conductor (table on page 178);

F = variable, whose value depends on the kind of system and the nature of the load; and

M = variable, which depends on the size of wire, the power factor, and the frequency.

Then the area of conductor $= \frac{DWK}{PE^2}$ in circular mils.

Volts lost in line $= 0.01\ pEW$.

Current in main conductors $= \frac{WF}{E}$.

Pounds of copper $= \frac{D^2WKA}{1,000,000PE^2}$.

EXAMPLE.—Find the size of conductor necessary to transmit 40,000 watts to a distance of 20,000 feet, the line voltage being 2000 and the permissible loss 10 per cent. Transmission is to be by the three-phase three-wire system, at 60 cycles, power factor of 85 per cent.

Area of conductor $= \frac{20,000 \times 40,000 \times 1500}{10 \times 4,000,000} = 30,000$ circular mils.

The nearest size, No. 5 B. & S. gauge.

Volts lost in line $= 0.01 \times 10 \times 2000 \times 1.06 = 212$ volts.

Current in main conductors $(I) = \frac{40,000 \times 0.68}{2000} = 13.6$ amperes.

Resistance.—In alternating current resistance is the same in kind as in continuous current, but is generally considered as negligible in comparison with other alternating-current factors. The effective value of an alternating current is 1.41 times its apparent value.

The Power Factor of an alternating-current circuit is the ratio of the kilowatts, as indicated by a watt-meter, to the apparent watts or volt-amperes. It enables one to readily determine the true energy in a circuit when the apparent energy is known; likewise, the resistance when the impedance is known; the energy E.M.F., when the impressed E.M.F. is given, and the energy current when the maximum current is known.

Electric Systems.—The available electric power systems comprise the continuous-current, the polyphase alternating current, and a combination of the two. When the machinery to be installed is merely traction and lighting, the continuous-current system will be employed. Being essentially a low-pressure current, it must be used locally and cannot be transmitted over great distances.

Either a two-wire or a three-wire line may be supplied by

	Values of A, or Circular Mils.	Values of K, Per Cent Power Factor.					Values of F, Per Cent Power Factor.				
		100	95	90	85	80	100	95	90	85	80
Single-phase..........	6.04	2160	2400	2660	3000	3380	1.00	1.05	1.11	1.17	1.25
Two-phase (4-wire)....	12.08	1080	1200	1330	1500	1690	.50	.53	.55	.59	.62
Three-phase (3-wire)...	9.06	1080	1200	1330	1500	1690	.58	.61	.64	.68	.72

VALUES OF *M* FOR WIRES 18 INCHES APART.

Size of Wire, B. & S. Gauge.	Area of Wire, Circular Mils.	Weight of Wire, Bare, per 1000 Feet (in Lbs.).	Resistance of Wire, per 1000 Feet at 20 C. (in Ohms).	25 Cycles, Per Cent Power Factor.				40 Cycles, Per Cent Power Factor.			
				95	90	85	80	95	90	85	80
000	211,600	640.73	.04879	1.23	1.29	1.33	1.34	1.52	1.53	1.61	1.67
00	133,079	402.97	.07758	1.14	1.16	1.16	1.16	1.25	1.32	1.35	1.37
4	41,742	126.40	.2473	1.02	1.00	1.00	1.00	1.05	1.06	1.03	1.00
5	33,102	100.23	.3120	1.00	1.00	1.00	1.00	1.03	1.01	1.00	1.00
6	26,250	79.49	.3934	1.00	1.00	1.00	1.00	1.01	1.00	1.00	1.00
7	20,816	63.03	.4958	For sizes 7 and 8 wire the constants are within .05% (average) of 5 and 6.							
8	16,509	49.99	.6250								

Size of Wire, B. & S. Gauge.	Area of Wire, Circular Mils.	Weight of Wire, Bare, per 1000 Feet (in Lbs.).	Resistance of Wire, per 1000 Feet at 20 C. (in Ohms).	60 Cycles, Per Cent Power Factor.				125 Cycles Per Cent Power Factor.			
				95	90	85	80	95	90	85	80
000	211,600	640.73	.04879	1.62	1.84	1.99	2.09	2.35	2.86	3.24	3.49
00	133,079	402.97	.07758	1.34	1.52	1.60	1.66	1.86	2.18	2.40	2.57
4	41,742	126.40	.2473	1.12	1.10	1.11	1.10	1.27	1.35	1.40	1.43
5	33,102	100.23	.3120	1.08	1.08	1.06	1.04	1.21	1.27	1.30	1.31
6	26,250	79.49	.3934	1.03	1.02	1.00	1.00	1.12	1.14	1.14	1.13
7	20,816	63.03	.4958	For sizes 7 and 8 wire the constants are within .05% (average) of 5 and 6.							
8	16,509	49.99	.6250								

the D.C. generators. With the latter, the generator must have three wires to deliver continuous current of two pressures, one being double that of the other. If this type of dynamo is not used, either two generators are installed in series or else some complex form of balancing device is required.

When the power is required for pumping, lighting, coal-cutting, etc., the alternating current can be more economically transmitted. Its machines are easier to construct than D.C. machines of similar pressures and its motors give less spark and

are of more general application. Its higher efficiency is due to the high voltage to which it can be carried. This is what gives the polyphase current its chief claim. Particularly is this true in mountainous districts, where sources of water-power distant from the mines can be developed for electric generation and thence distributed to various points.

In the event that both systems must be maintained in the mine, both currents may be generated and conducted, or it is practicable to produce the cheaper polyphase alternating current, and, by the use of converters at suitable points, to change only such portion of the energy into direct current as will be required for the haulage and lighting, leaving the remaining portion of the current for the other mining operations.

Converters.—Changes from the alternating current to a direct current, or the reverse, can be effected by the means of a rotary converter. Changes from a high-pressure to a low-pressure current, or the reverse, whether D.C. or alternating, may be effected by a transformer for stepping down or stepping up in pressure. As reliability is of the utmost importance, transformers are always installed of twice the capacity for the expected service.

Their efficiency is about 96 per cent.

The Transmission Difficulties.—The great objection to electricity is the high cost of wire and erection. Economy of copper wire is possible only by employing high voltage in the currents, as shown by the formulæ already discussed. As an example, 195 lbs. of copper can transmit 50 amperes at 2000 volts over a 5000-foot line with the same percentage of line loss that 795 lbs. of No. 3 B. & S. gauge can transmit the same quantity at 500 volts. But in the transmission other difficulties are introduced. The insulation must be perfect and the wires separated from one another sufficiently to prevent an induction of one current upon the other. From both sources the line loss would be large. With proper insulation, however, the limit of voltage which can be carried would be fixed by the distance between the wires. There must be a wide gap between them, for, with the enormous voltage which is now being used, we are approaching

the conditions in lightning, not in a single stroke but a number of strokes following each other in very rapid succession. The length across the air-space increases as the voltage increases, and for currents of 50,000 volts the arms carrying the wires at their ends must be 12 feet long.

There is no limit to the E.M.F. which the alternating-current machines can impart to the current. The majority of long-distance transmission companies employ the three-phase system and at as high a difference of potential as possible. A high-voltage current may be obtained either directly at the dynamo whence it is delivered to the distant end, or the dynamos may generate a low-tension current which is transformed into the high-voltage current at the dynamo station for transmission to the distant point. At the mine it may be transformed wholly into alternating current of tension suitable for the machines, or only in part, the balance being converted to a continuous current for haulage.

The Most Economical Area of Wire.—The losses in transmission are due either to resistances or to leakage. The drop in pressure caused by leakage of the current from the wire through the points of support is deduced by insulating the supports and an enveloping tube of non-conducting material around a rubber-covered wire. The drop in voltage, due to interaction of neighboring parallel circuits, is averted by an ample air-gap. That due to the resistance of the wire to the passage of the current is controlled by the use of a larger wire. The resistance of the wire varies inversely as the area of its cross-section. A small wire offers a great resistance to the electric fluid. This loss reduces the available pressure at the terminals, and consequently the amount of power there obtained. It develops heat in the conductor, which is not only injurious to the wire, but, without secure fire protection, is also dangerous to the surrounding medium.

In calculating the dimensions of the conducting wire a balance is made between the elements—great first cost of a large quantity of copper and the waste of power during the life of the plant by use of small wire. The allowable expenditure of wire, in any

given plant, will determine the E.M.F. which can be used; the efficiency required of the plant will fix the allowable line loss. Hence the most economical area of conductor is that for which the annual interest on the capital outlay equals the annual cost of the energy thus wasted.

The Allowable Voltage in Mines.—There are many conflicting requirements for electric wiring in mines. The great distances of distribution require high voltage to save copper. This demands a high degree of insulation, which is difficult to maintain in the moist air and the mine-gases. Moreover, the conditions of work require bare wires, and their position is such as make them dangerous to life. The porcelain or glass tubes are convenient insulators, but they do not prevent shock.

The lowest difference of potential causing fatal accidents is 220 volts with a direct current and 110 volts with an alternating current. The more the difference of potential increases beyond these figures the more must all tensions be regarded as equally dangerous. Though a higher voltage would be more economical in the cost of copper for a given power transmission, the miner is restricted to a low voltage because of the risks imposed by fire and shock. The allowable maximum voltage has been fixed at 450.

In the three-wire system one trolley wire is used on each side of the traction generator or to the free sides of two generators in series. In the former case the conductors from the rails should connect with a small dynamo to keep the rail pressure to one-half that of the main terminal. In the latter case all rail conductors are taken to a common point of connection. By grounding the neutral wire the shock at contact would not be so serious unless both wires were simultaneously touched. The trolley circuits, however, subject the maker of the contact to the full discharge of pressure between the two sides of the three-wire system. So, too, a ground connection of the high-pressure circuits, or a cross between the high- and low-pressure circuits, inside the transformer, would subject him to fatal results.

For traction a pressure of 250 volts is quite sufficient, even with an induction motor, for it can be operated by transformers

at every few hundred feet to keep the voltage down. Incandescent lamps on a two-wire D.C. system are operated on 125 or 250 volts, and at double these pressures on the three-wire distribution system. On alternating-current circuits the lamps are supplied by near-by transformers. Lamps and stationary motors are carried under a maximum pressure of 250 or 500 volts, with corresponding differences of pressure between any wire and the earth of one-half the quantity.

The Motor.—A motor converts the current into mechanical energy. It is directly connected, or is belted, to the mechanism to be operated. The construction of the motor is practically the same as the dynamo, but its function is exactly the opposite. The current flows in the reverse order, passing through the coils of the armature and the field-magnets. These react upon the coils, carrying a current into the armature and thus produce rotation. The force turning the armature is called its torque and is its capacity to do work. Around the field, magnetized by the current flowing around the magnets, a current, called the back E.M.F., is induced in the opposite direction. This increases with the current intensity and the number of coils and their speed. Taking current in proportion to the work performed, the motor is self-acting.

The continuous motors are wound in three ways, corresponding to those of the dynamo, each having its own field of usefulness. In the series motor there is but one continuous circuit through the stationary and rotary pieces. This exerts a greater starting torque, but its speed will change with every variation of the load. It is intended for constant load only. If it is overloaded, its speed is slackened. Its torque is increased by its greater draft of current, and if it is overloaded, it runs slower, the field-magnets heat up and burn out, or a fuse blows. Removing the load suddenly causes it to race, because the field is weakened and the back E.M.F. is diminished. The increased speed which ensues may ultimately injure the motor. The shunt motor will maintain a constant speed and a back E.M.F. even with variations of the load, but its starting power is low. Designed

for the maximum load which it is expected to drive, it is employed in mining work. It is heavier than the series motor. The compound motor is used about the mines and combines the shunt and the series features of motors.

Alternating-current Motors.—These motors are either synchronous or non-synchronous. The speed of the former type bears a constant relationship to the frequency of the driving current. Being unable to start until it synchronizes with that of the generator, miners use it very little.

In the non-synchronous motor, also called an induction motor, the speed varies with the load and is independent of the frequency of the current. It is self-starting. Induction motors may be obtained of the two-phase four-wire type, or the polyphase type with three wires. In the two-phase motors each coil of a set in the rotor receives its own current. One induces a current in the conductors of the rotor, causing attraction between them and the stator coils. The slight rotation that follows changes the current to the next set. Each set of windings produces similar results and adds to the rotary effort. In the three-phase system continuous rotation occurs under similar conditions, except that the entering current is divided among three separate sets of coils. A back E.M.F., as an opposing pressure, is induced in the conductors of the rotor, as in the D.C. machine. When the load is increased the speed falls, the opposing pressure becomes less. There is an increase in the available driving pressure, or torque, and an increase of current in the rotor. This alters its speed relative to the rotating field. Every alteration of current in the rotor conductors reacts on the stator coils and reduces the strength of the magnetic field, until the energy supplied to the motor is insufficient for the work and the motor stops. In this respect the induction motor differs from the D.C. motor. An increase of load brings the former to a standstill, but results in the burning of armature coils in the latter. An induction motor can take an excess of 20 per cent over its normal load without injury. Having neither commutator, induction brushes, nor movable contacts, it is the ideal motor for mining machinery where the

air may be laden with dust, gas, or moisture. The power factor is over 90, with an efficiency proportionately high.

The objections urged against the polyphase induction motor are its excessive starting current, which results in a drop of voltage with consequent bad regulation to other apparatus in current, and its inflexibility in the matter of speed regulation. The first objection against the short-circuited secondary type is negligible if the power be increased at the generating station on starting. The machine is essentially a constant-speed motor. When variable speed is required it would be more advantageous to employ a variable resistance with the secondary type of induction motor effected by a rheostat.

Rheostats.—There is no back E.M.F. in the circuits at the moment of starting a motor, and, to prevent the heavy entering currents from doing damage to the armature, extra resistances must be introduced into the current until such times as the motor has got up speed. The starting resistance consists of coils of German-silver or iron wire contained in fire-proof boxes. A movable contact, turned to one side or the other from its central position, determines the direction of rotation. It is operated by an attendant. On closing the circuit the current flows through the entire resistance of all the coils. As the motor speeds up, one after the other of the coils in the circuit is gradually cut out. The diverter of columns of iron and mica in five separate portions is used for locomotives.

The resistance, on shunt- and compound-wound machines, is placed in the shunt winding of the fields to increase or decrease the E.M.F. There is usually on pump-motors an additional safeguard of an automatic release for the rheostat contact lever. The contact arc is worked against a spring, and is held, when fully on, by means of a magnet, which is connected in series with the shunt fields of the motors.

For alternating-current machines the starting resistance consists of two rings connected to the conductors and insulated from one another and from the motor shaft. Fitted to them are brushes with a resistance inserted between them, which can be automatic-

ally cut out as the motor attains its speed. These slip-rings are covered in mining machines to confine the spark. Another class of starter is that in which the current passes through water into which is lowered a lead cone from the surface to the bottom where the other terminal is located. This form of switch is free from spark, though it might be dropped too quickly and thus be almost valueless.

The variable speed induction motor has a two-drum controller for resistance and reversing operated by a single handle. The resistance is in series with the rotor windings, and is cut out in the usual way until the approximately constant speed is attained.

Reversing-motors.—When the direction of the motor is to be reversed, its brushes should bear vertically on the commutator and a reversing-switch be used. This appliance for changing the direction of the current in the field-magnets is arranged as a part of the starting resistance.

Enclosed Motors.—If possible, motors should be run open, for if totally enclosed a much larger motor will be required for the same work on account of the difficulty of dissipating the heat developed in the coils. If precautions are observed not to overload the machine, the danger is reduced and certainly would not be greater than that existing from either cause in the daily operations. They are rarely placed near gaseous mixtures, and the danger of igniting neighboring timbers by their spark is very slight. For safety in mines, however, electric motors are enclosed to protect them from fall of rock, dust, and dampness.

The Efficiency of a Motor.—Not all the current which the motor receives is usefully applied. Part of it is spent in heating the circuit of the motor and part in overcoming frictional resistance. The remainder, which is available for mechanical work, is still, however, quite large. As large motors have high efficiency, it is possible to attain very high results, if their capacity be large enough. This is, however, a question of weight and first cost. The efficiency of a machine is not far from 90 per cent under average conditions, and will reach 92 per cent when operating under the load and speed for which it was designed. The

table below gives the number of watts necessary to produce one effective horse-power at the terminals of a motor with stated commercial efficiencies. Thus a motor with 10 effective horse-power having an efficiency of 85 per cent will require 8780 watts of current.

Efficiency of a Motor.	Watts to be Delivered to it per Horse-power.
1.00	746
0.98	764
0.95	774
0.90	830
0.85	878
0.80	933
0.75	1000

EXAMPLE.—The efficiency of a 50-H.P. motor is 0.90. It is to be driven a half mile from a dynamo whose efficiency is 0.85. The modulus of the engine driving the generator is 0.80. Required th horse-power of the engine. Volt ge, 400.

The motor requires 830 watts per horse-power or 41,500 watts total:

$$41{,}500 \div 400 = 103.75 \text{ amperes.}$$

For this the wire must be No. 4. Its resistance, per thousand feet, is 0.245 ohms.

The watts lost per thousand feet are

$$I^2R = 104 \times 104 \times 0.245 = 2650.$$

For the mile of line in the circuits 13,992 watts are lost.

The dynamo must therefore furnish

$$41{,}500 + 13{,}992 = 54{,}492 \text{ watts.}$$

This corresponds to a voltage at the dynamo of 533, and a loss of 33 per cent.

$$\text{B.H.P} = \frac{55{,}492}{746 \times 0.85} = 87.6 \quad \text{and} \quad \text{I.H.P.} = \frac{87.6}{0.80} = 109.5.$$

A No. 2 wire, B. & S. wire, will show a loss of 20 per cent, a dynamo voltage of 481, and an I.H.P. of 100.

Water-power.—In the immediate vicinity of the wheel water the power has been employed for various operations, but not until the modern successful utilization of electricity has a cheap efficient means been discovered for its transmission to great distances. The installation of electric plants is opening the possibilities of water-power to an enormous degree.

The gross power possessed by water and the potential energy which is capable of being transmitted and converted into kinetic energy is measured by the product of the weight of water discharged by the height, h, through which it has fallen. Assuming the weight of a cubic foot of water at 62.5 and Q as the number of cubic feet discharged per minute, then a horse-power equals $0.00161Qh$. The net power obtained from this in effective work is from 40 to 90 per cent according to the character of wheel used.

This energy develops power by driving one of several types of wheels, of which the undershot and the overshot were the earliest. They consume, however, enormous volumes of water and are used only for small machinery. The turbine-wheels, placed horizontally, revolving under a pressure due to a head of water above, give a high efficiency with the use of a large volume of water. These wheels are not very large and are encased in a globe or cylindrical casing above which is a penstock through which the water flows. The water enters centrally and discharges circumferentially and produces a high rate of revolution. The heads at which these turbines are used rarely exceed 200 feet.

Impulse Wheels.—The more recent forms of wheels are of the impulse type, in which a small wheel receives at its circumference the impact of a stream flowing with a high velocity due to a head which in some cases is as great as 1900 feet. The periphery of the wheel carries a number of small cup-shaped vanes whose curves and position are such as to receive the full impact of the stream and to discharge the water at a velocity of nearly zero. The efficiency of these wheels is therefore high, and their construction is very simple; the inferior limit of head with which they may operate to advantage is about 30 feet. The maximum limit of head is determined only by the strength of the material in the wheel. As is the case with the De Laval steam-turbine (Fig. 48), multiple jets are also used on water-turbines.

The diameter of the wheel is proportioned to the rate of

revolution desired for the main shaft. Usually the latter is that for which the electric generator has been wound and designed. The diameters vary between 18 and 90 inches.

There are several types of impulse motors in America, of which the Doble, Pelton, and Knight are excellent patterns. The efficiency of these wheels when properly regulated by governors is 85 per cent of the theoretical head, due to the velocity of discharge from nozzles against the wheel-cups. In the mountainous districts, where the numerous creeks do not carry much water but have a large fall, these wheels may be used to advantage by laying a pipe-line from the wheel to some elevated reservoir in the creek constructed for the purpose. Not infrequently two or even three nozzles feed the wheel when the quantity is large enough to permit it.

The supply of water delivered to the wheel is usually estimated in miners' inches, representing a flow of something near 1.5 cubic feet per minute. The miner's inch is known as the volume of water which can be discharged through each square inch of an aperture 2 inches high and 4 inches long which is cut through a plank 1.25 inches thick, the lower edge of the aperture being 2 inches above the bottom of the measuring-box and the upper edge 5 inches below the level of the water.

Flow of Water through Pipes.—Owing to the roughness of the internal surface of a pipe and the restrictions which occur at elbows, joints, and valves, the actual velocity of discharge of water from a pipe is not equal to that due to the pressure at its inlet. Any change of area or of direction affects the flow of the water and produces frictional resistances which are directly proportional to the length, inversely as the diameter, and increase with the velocity. Though a theoretical formula may be evolved which expresses the relation between the discharge of water and the head producing such discharge, empirical formulæ are employed with coefficient inserted to provide for the various elements.

The formulæ below are given to assist the engineer in determining the dimension of a pipe requisite for the given flow of water with a stated resistance.

Fluid Friction in Pipes.—Assuming very long clean pipes of uniform size, the resistance which is afforded by the interior surface of the pipe to flow is measured in terms of the head.

Fig. 72.—Direct-connected Electric Generator and Impulse Wheel.

Let Q = the quantity of water discharged, cubic feet per second;
v = the velocity in feet per second;

d = the diameter of the pipe in feet;
l = the length of pipe in feet;
f = the coefficient of friction;
H = the gross head of water—the difference in elevation between the two termini of the pipe;
h = the frictional loss measured in terms of feet of head, and
h_1 = the effective head $= H - h$.

In clean pipes of smooth bore, $f = 0.004$ and in ordinary mine pipes $f = 0.006$.

$$v = 1.273\frac{Q}{d^2}.$$

$$v = 203\left(\frac{d}{4}\right)^{0.694}\left(\frac{h}{l}\right)^{0.555};$$

$$h = 0.0006061\,l\frac{Q^{1.802}}{d^{4.855}};$$

$$d = \left[0.0006061\,l\frac{Q^{1.802}}{h}\right]^{0.206}.$$

Q, for the maximum horse-power,

$$= H^{0.555}d^{2.694}\frac{1}{0(.0017l)^{0.555}}.$$

The maximum horse-power to be obtained is equal to

$$\text{H.P.} = 2.466H^{1.555}\left(\frac{1}{l^{0.555}}\right)d^{2.694}.$$

In Fig. 73 are curves based upon the above formulæ, giving the loss of head due to the frictional flow through pipes which are ordinarily used in pumping and for transmitting power to impulse wheels. On the right-hand side will be found a vertical line giving the theoretical horse-power necessary to raise a given quantity of water indicated on the left vertical line through a height of 100 feet. Thus, raising 7 cu. ft. per second through any pipe for a height of 100 feet requires 80 horse-power theoretically. When the height is other than 100 feet the result obtained from the diagram is a multiple of the 100-ft. eleva-

tion. To raise the same quantity of water in the same pipe through 600 feet of pipe will therefore require ($80\times6=$) 480 horse-power.

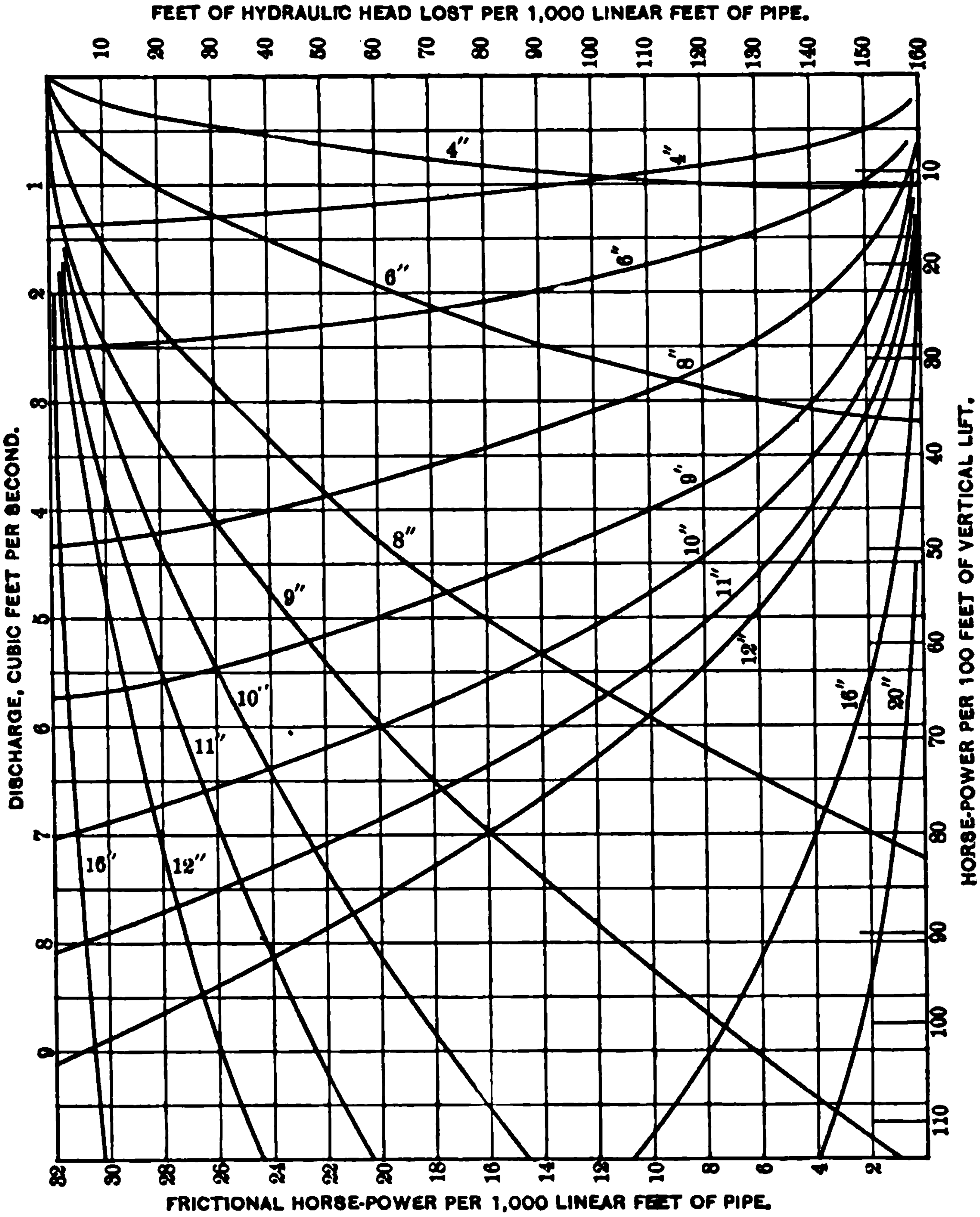

FIG. 73.—Loss of Hydraulic Head in Pipes.

The horizontal line at the top gives the amount of head lost by friction, and the horizontal line at the bottom the horse-power needed to overcome the friction. Thus let it be required to find the head lost and the horse-power while raising 7 cu. ft. per

second in a 12-inch pipe 5000 feet long on a slope of 12 per cent. The curve shows that 1000 feet of 12-inch pipe carrying this volume of water consumes 20 feet of head. Thus 5000 feet will consume 100 feet of the head. Likewise 1000 feet will consume in this lost head about 28 horse-power, while 5000 feet will consume 140 horse-power in overcoming friction.

The frictional loss in a 10-inch pipe 1000 feet long, carrying 2,100,000 gallons per day (3.245 cu. ft. per second), is 12.8 feet. An 8-inch pipe would have consumed 38.5 feet of head under the same conditions of length and discharge.

EXAMPLE.—A pipe 500 feet long is to deliver 4 cubic feet per second. Required its diameter if its frictional loss of head allowed is 30 feet. For 30 feet hydraulic head per thousand feet of pipe and 4 cubic feet of flow the diameter should be 9 inches. 500 feet of pipe consuming 30 feet will correspond to a length of 1000 feet consuming 60 feet of head. For this condition and 4 cubic feet of discharge the pipe may be of 8 inches diameter.

Kutter's Formula for Ditches and Flumes.—Fig. 74 contains curves plotted according to the Kutter formula, which is acceptable for all ditches and flumes of medium dimensions. The formula is based on the average condition of the frictional surface. The upper curves are for flumes with hydraulic radii of from 1 to 9. The coefficient of roughness, N, for flumes is taken at 0.011, and for ditches at 0.026. On the left are laid off the gradients in terms of length of line giving a fall of 1 foot, and the computed velocity of flow is on the top line. Then the four lower curves on the left represent the relation between velocity and grade in ditches having hydraulic mean radii, H.M.R., of 0.5, 1, 2, and 3, respectively. By the hydraulic mean radius is understood the quotient obtained by dividing the cross-sectional area of the ditch by the length of the watered surface of the ditch.

EXAMPLES.—1. What would be the loss of head in pumping 2000 gallons of water per minute through an 8-inch pipe 600 feet high? $Q=4.44$, and $h=47.6$, or 38.3 feet, according to the equations employed.

2. What horse-power is consumed in overcoming friction in the previous example? Assume h to be 38.3. 19.4 H.P.

The flow is 4.44 cubic feet per second, and the horse-power is $0.1134\ QH$.

3. What horse-power will be given out by the discharge of 400 cubic feet of water per minute from a pipe of 13 inches diameter, 2600 feet long, with a head of 400 feet?

The loss of head is 32.53 feet, and the horse-power available is 276.

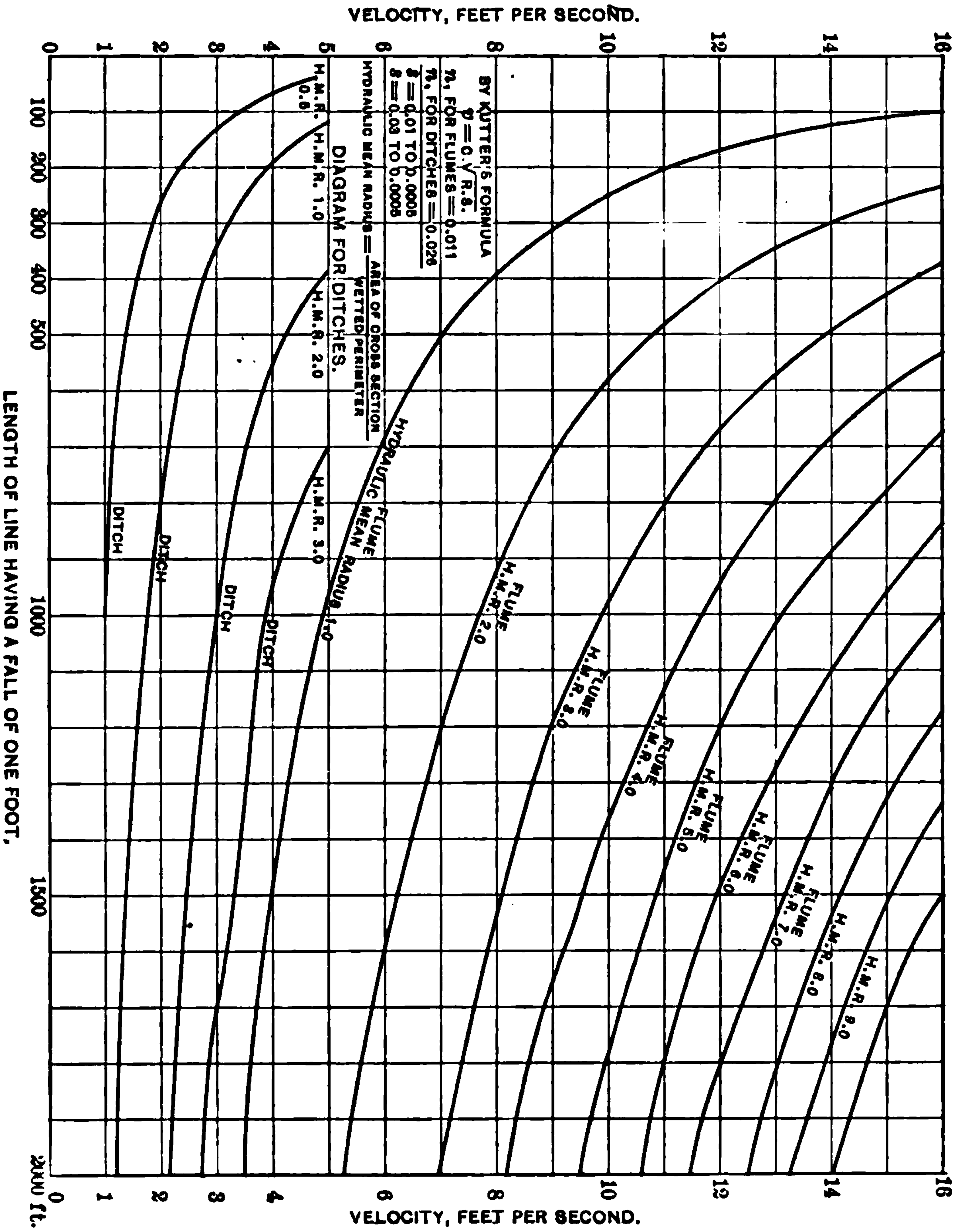

FIG. 74.—Flow of Water in Flumes and Ditches, according to Kutter's Formula.

4. Let it be required to determine the flow in a trapezoidal ditch whose area is 72 square feet and whose wetted perimeter is 24. The H.M.R. is therefore three. If the fall of the ditch is 1 foot in 1500 feet, the velocity of the

flow is 3.5 feet and the discharge is ($72\times 3.5=$) 252 cubic feet per second. What should be the grade of a flume $12'\times 6'$ which has a H.M.R. of 3, when the quantity desired is to be 547 cubic feet per second? This corresponds to a velocity of discharge 7.58 feet per second.

In the event that a ditch under computation has a mean hydraulic radius other than that platted among the curves it may be easily interpolated. Thus one of an h.m.r. of $1\frac{1}{2}$ on a grade of 1 foot in 400 feet will be found to have a velocity of about 3.8 feet per second.

5. A pipe is 500 feet long and 3 inches diameter. What should be the head to produce a discharge of 180 feet per minute?

Here $Q=3$, $1=500$, $d=0.25$, and assuming f to be 0.00566,

$$h=0.1007\times 0.00566\times 500\times 9\times 1024=2624 \text{ feet.}$$

$$H=h+h_1=2624+\frac{V^2}{2g}=3124 \text{ feet.}$$

6. What diameter should it have to deliver the same quantity of water with a head of 82 feet?

7. Required the flow of water through a pipe 2000 feet long, 13 inches in diameter, and 200 feet head. For the maximum horse-power we have

$$(200)^{0.555}(1.08)^{2.694}\frac{1}{(3.4)^{0.555}}=0.864 \text{ cubic feet per second.}$$

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CHAPTER VII.

HOISTING MACHINERY AND UNDERGROUND CONVEYANCES.

Underground Conveyances.—Where possible, the mineral is loaded at the face into cars or buckets to avoid frequent handling. But in metal-mines and in steep-pitching coal-veins this is not possible, and the product is delivered through shutes, or batteries, to the cars at the lower level.

Where the output is small and the depth of shaft is slight, metal-mines employ buckets which are frequently drawn through the haulage-way on the trucks, and thence hoisted to the surface to be dumped. All coal-mines and the larger metal-mines employ cars as the common carriers for the mineral. They are loaded as stated and drawn through the haulage-ways to the foot of the shaft, or slope, through which they are hoisted to daylight.

Occasionally when the pitch of the slope exceeds 40° the cars are emptied at the level landing into skips, which are hoisted to the surface, while the car returns to its loading station. Slope carriages are also used, upon which the car is delivered and drawn to the surface to be emptied. In vertical shafts the car is hoisted to the surface on a cage, and returned underground when emptied.

The several conveyances named will be here discussed together with the hoist-rope, signalling and safety devices, and the method of dumping at the surface.

Hoist-ropes.—Hemp, aloe, fibre, and iron or steel wire are used for hoisting purposes and for the transmission of power in

mines. They are wound into strands and ropes to furnish a strong, flexible, light means of haulage. The hempen ropes for hoisting purposes have been supplanted by the flexible steel wire for many reasons, prominent among them being the great weight of the former compared with its strength and the unreliability of its fibres. So frequently are the hempen ropes built up of strands of discarded ropes that long service is never obtained from them. This fact, together with their absorption of moisture while in use, contributes to their rapid decay.

Steel-wire Ropes.—The successful production of fine iron or steel wire and its conversion into flexible ropes for hoisting has led to the abandonment of various vegetable fibres. The usual number of strands in a multi-wire rope is six, with nineteen wires in each strand, or sometimes only seven wires are used to the strand. The latter are more frequently employed for guide-ropes than for motor purposes, being less pliable than the former.

The increased strength imparted to metallic wires by drawing, and the high grade of steel wire as compared with iron, at once recommend it for any purpose where strength and lightness are requisites. Good crucible-steel wire has an ultimate breaking strength of 75 to 100 tons per square inch. Tempered steel, usually known by the indefinite name of plow-steel, is a high-grade cast steel, used for ropes of the greatest tensile strength.

Standard Ropes.—Hoisting-ropes in the United States are made with hemp centres, around which are twisted six strands of nineteen wires each, the individual wires being about one-fifteenth the diameter of the rope. The hemp centre imparts flexibility and furnishes a soft yielding core upon which, with a minimum of injury or friction, the strands may "creep" as the rope passes around the drum and sheave. When great flexibility is required, each strand has also its own centre, but this is unnecessary for ordinary hoisting service. On account of the hardness and diminished flexibility of plow-steel wire, hoisting-ropes of this metal are often made with eight instead of six strands, the

wires being of smaller gauge and therefore more flexible. Though the enclosing figure of the hoisting-rope is a polygon, the rope is usually called a round.

The Lay of the Rope.—By this is meant the twist or pitch of the wires in the strands, and of the strands in the rope. As ordinarily constructed, the lay of the wires is opposite to the lay of the strands. The former is as long as possible in order that the frictional wear on each rope shall be distributed over a great length rather than to have a short twisted strand on which the wear is concentrated. The so-called Langlay ropes have a longer twist to each wire, thus distributing the wear over a greater length and giving a smoother surface.

Locked-coil Ropes.—In order to increase the smoothness of the rope and thus to reduce wear, ropes are frequently made of wires which are locked together or dovetailed. The locked-coiled rope is of this type, and prevents the ends of the broken wires from protruding. Its wires are built of a specially shaped cross-section and wound in concentric layers, the successive layers being twisted in opposite directions. The exterior layer presents a perfectly smooth surface.

Flat Ropes are employed with winding reels where it is desirable to aid the engine without the use of a long conical drum. Moreover, the tendency to uncoil which exists in the several strands and wires of round rope develops a whirling tendency upon the cage as it travels in the shaft. This the flat rope prevents. Flat ropes are heavier than a round rope of the same strength, but are of shorter life, costing more and requiring greater care. A flat rope is composed of an even number of loosely twisted four-stranded round ropes, without hemp centre, laid side by side and sewed through the wires. The lay of any two adjacent strands is in opposite directions. The sewing wires, from eight to twelve in number, are passed from side to side through the centres of the strands. Flat ropes range in width and thickness from $2\frac{1}{2}'' \times \frac{3}{8}''$ to $9'' \times \frac{7}{8}''$.

Although the breaking strength of a flat rope is nearly the same as that of a round rope of equal weight per foot, its safe

working load is considerably less, due to the practical impossibility of so constructing the rope that all the strands will stretch uniformly and be equally loaded in service.

Taper Ropes.—As the depth of shafts increases, the use of round wire rope of uniform cross-section is attended with many difficulties. The diameter of the rope becomes greater, the inequality of the work upon the engine is more marked, and an excessively large of drum will be required. There is a limit beyond which a rope of uniform section cannot safely carry its own weight. At 12,000 feet of depth, exclusive of any live load, the rope has reached the limit of its safe working stress. The additional weight of car and load reduces this materially. With the margin provided for safety, the maximum depth is 3000 feet. A given rope for carrying men having a factor of safety of 15 when less than 1000 feet deep has a factor of but 10 at 2500 feet and only 7 at 3000 feet.

Meanwhile the cross-section of the rope at the bottom is excessively heavy, having little load to support. If the cross-section be such as will support the live load, cage, and car, then the cross-sections of the rope to the top may increase in area by an amount equal to that sufficient to support the additional weight of rope below that point. Hence tapering ropes are employed in special cases, being of uniform strength, with such a factor of safety as may be desired. Theoretically, the use of a tapering rope increases the limit of depth which can be reached within mining possibilities, but practically it is not a popular hoister. The round tapering rope is made by discontinuing at intervals, throughout the length, a single wire at a time. The cross-section of the rope is uniformly decreased toward the lower end, the thickness at any point being sufficient to carry safely the load at that point. One round taper rope is of steel of five sections, varying in diameter from $1\frac{1}{8}$ to $\frac{15}{16}$ inch. The sections are spliced together and the ends of the wires soft-soldered. In some European districts flat taper ropes of Manila fibre are still employed for deep shafts.

The taper rope is not well adapted to long slopes, for its lower

portion, which is the smaller end, traverses a greater distance than any other portion of the rope and is subjected to the greatest amount of abrasion. This is the reverse of the situation in a shaft, where the thickest part of the rope has the wear.

The Strength of a Wire Rope.—This is greater than that of a steel rod of equal cross-section and same material, because of the increased strength imparted to wires during the process of drawing; but it is not equal to the sum of the strengths of the individual wires.

The approximate weight of a wire rope in pounds per foot is ascertained by multiplying the square of its diameter in inches by 1.58

CAST-STEEL HOISTING-ROPES:

THEIR ULTIMATE STRENGTH, MAXIMUM SAFE STRENGTH, AND WORKING LOAD.

Diameter in Inches.	Approximate Circumference in Inches.	Estimated Weight per Foot in Lbs.	Proper Working Load, L.	6 Strands, 79 Wires each.		6 Strands, 19 Wires each.	
				Ultimate Strength, Lbs.	Maximum Safe Stress, S.	Ultimate Strength, Lbs.	Maximum Safe Stress, S.
1 3/8	4 1/4	3.00	Maximum safe stress less bending stress.	116,000	38,667	124,000	41,333
1 1/4	4	2.45		96,000	32,000	100,000	33,333
1 1/8	3 1/2	2.00		80,000	26,666	84,000	28,000
1	3	1.58		64,000	21,333	68,000	22,000
7/8	2 3/4	1.20		48,000	16,000	52,000	17,333
3/4	2 1/4	0.89		37,200	12,400	38,800	12,933
5/8	2	0.62		26,400	8,800	27,200	9,067
9/16	1 3/4	0.50		21,200	7,067	22,000	7,333
1/2	1 1/2	0.39		16,800	5,600	17,600	5,867
7/16	1 1/4	0.30		13,200	4,400	13,600	4,533
3/8	1 1/8	0.22		9,600	3,200	10,000	3,333
5/16	1	0.15		6,800	2,267	6,800	2,267
1/4	3/4	0.10		5,600	1,867	4,800	1,600

The Minimum Radius of Curvature.—The minimum drum diameter admissible for ordinary service is from 50 to 100 times the diameter of the rope. By using the larger ratio the average life of hoisting-ropes would be materially lengthened. The English make the sheave diameter a multiple of X^2. The length of the arc of contact between rope and sheave and the angle between the two branches of rope are not the same as would be

the case with a belt or Manila rope. A large part of the total strength of the rope is consumed in the mere strain of bending, and this is wholly a question of radius of curvature, depending upon the diameter of the sheave.

This bending resistance increases the tension on the wires of the ropes and reduces the available load which it can safely carry by that amount. The continual bending naturally reduces the life of the rope.

In the following table are given the values of the radii of curvature, R, in inches, assumed by new steel ropes of stated diameters having a tension of one pound upon them and having an angle of deflection or of bending of 1°.

THE RADIUS OF CURVATURE R OF STEEL-WIRE ROPE, IN INCHES, HAVING 1° DEFLECTION AND 1 LB. TENSION.

Diameter.	7-wire.	19-wire.
$\frac{1}{2}$	257,270	84,210
$\frac{5}{8}$	592,970	220,950
$\frac{3}{4}$	1,241,510	454,120
$\frac{7}{8}$	2,315,930	860,400
1	3,971,290	1,430,850
$1\frac{1}{8}$	6,386,740	2,246,670
$1\frac{1}{4}$	10,049,650	3,369,820

To obtain the radius of curvature under similar conditions for any other angle of deflection, it is only necessary to divide the corresponding quantity by the number of degrees of bend. Thus a new 1-inch 19-wire rope, for a tension of one pound upon it, would have a curvature whose radius is 1,430,850 inches when bent through an angle of 1°. If, however, it is bent 90° over a given sheave, the radius of the curve which it will assume then becomes 15,898 inches. When the tension becomes 1000 lbs. the curvature which it will assume will have a radius of 16 inches.

Bending Rope Around Curves.—When a rope is carried over a sheave with an angle of contact or wrap exceeding 30°, the diameter of the sheave must be of liberal dimensions. When, however, the rope is slightly deflected by a pulley or sheave,

the stress put upon the rope in bending is small. When it is bent over a succession of sheaves at close intervals, as in carrying a rope around curves in a roadway, the aggregate bending stress may become significant. The rope assumes between each pair of sheaves a curvature depending upon the tension it carries and the amount of each deflection.

FIG. 75.—Turning the Haulage-rope at Curves.

The distance between the sheaves may be determined from the formulæ below when the tension is known and the radius of curvature has been ascertained by the preceding table.

Let R = radius of curvature 1° bend with 1 lb. tension for rope of diameter X (from the preceding table);
r = radius of curvature assumed by rope under tension T;
d = diameter of individual wires in rope, inches;
n = number of wires in rope;
T = tension on the rope, lbs.;
s = distance between sheave centres, inches;
α = angle of deflection over a pulley, and
ρ = radius of centre line of sheaves, inches.

Then

$$r = 5,427,625 \frac{d^4 n}{T \cos \frac{1}{2}\alpha}.$$

The sheaves should be laid on a curve of radius r—that

is, $r=\rho$; $d=0.111X$ for a 7-wire rope and 0.066 for a 19-wire rope.

$$\alpha=\frac{R}{11{,}325{,}900nd^3};$$

$$s=2\rho\sin\tfrac{1}{2}\alpha.$$

The Influence of Bends on the Durability of a Rope.—The diameter of the sheave over which the rope is bent must in all cases exceed the minimum diameter of curvature assumed by the rope under the conditions as calculated above. Not only is the life of the rope increased, but also its bending stress is considerably diminished and the safe working load of the rope correspondingly increased. Under ordinary conditions of wear, the life of a rope for mining purposes averages about seventeen months, after which time the abrasion of some wires and the breaking of others render it unsafe. This assumes proper care in lubrication, in testing, care during hoisting to avert shocks, and the use of liberal-sized sheaves. Tests upon wires of various sizes passing over pulleys of various dimensions have demonstrated that a wire subjected to repeated bending soon breaks, the time of rupture being hastened as the diameter of the sheave or the angle of the bend is small. For example, a wire of No. 20 B.W.G., subjected to repeated bending over a 5-inch pulley, broke after making 15,200 turns; one of the same size and presumable strength made 453,000 turns over a pulley of 24 inches in diameter before breaking.

EXAMPLE.—It is desired to carry a $1\frac{1}{4}$-inch 19-wire steel rope around a curve (Fig. 75) whose track radius is 50 feet, by guide-sheaves 44 inches inside of the track centre. What should be the distance between the sheaves? $T=660$, $X=1.25$, $r=1{,}430{,}900$ inches, and $\rho=554$ inches. Then

$$a=3^\circ\ 00' \text{ and } s=2 \text{ feet } 5 \text{ inches.}$$

The Working Load.—This is a fraction of the ultimate strength of the rope, depending upon the margin of safety desired. For ordinary conditions of hoisting the factor of safety is taken at 7 for hoisting-ropes and 6 for those employed on slopes. The maximum stress to which the rope can be subjected should be less than

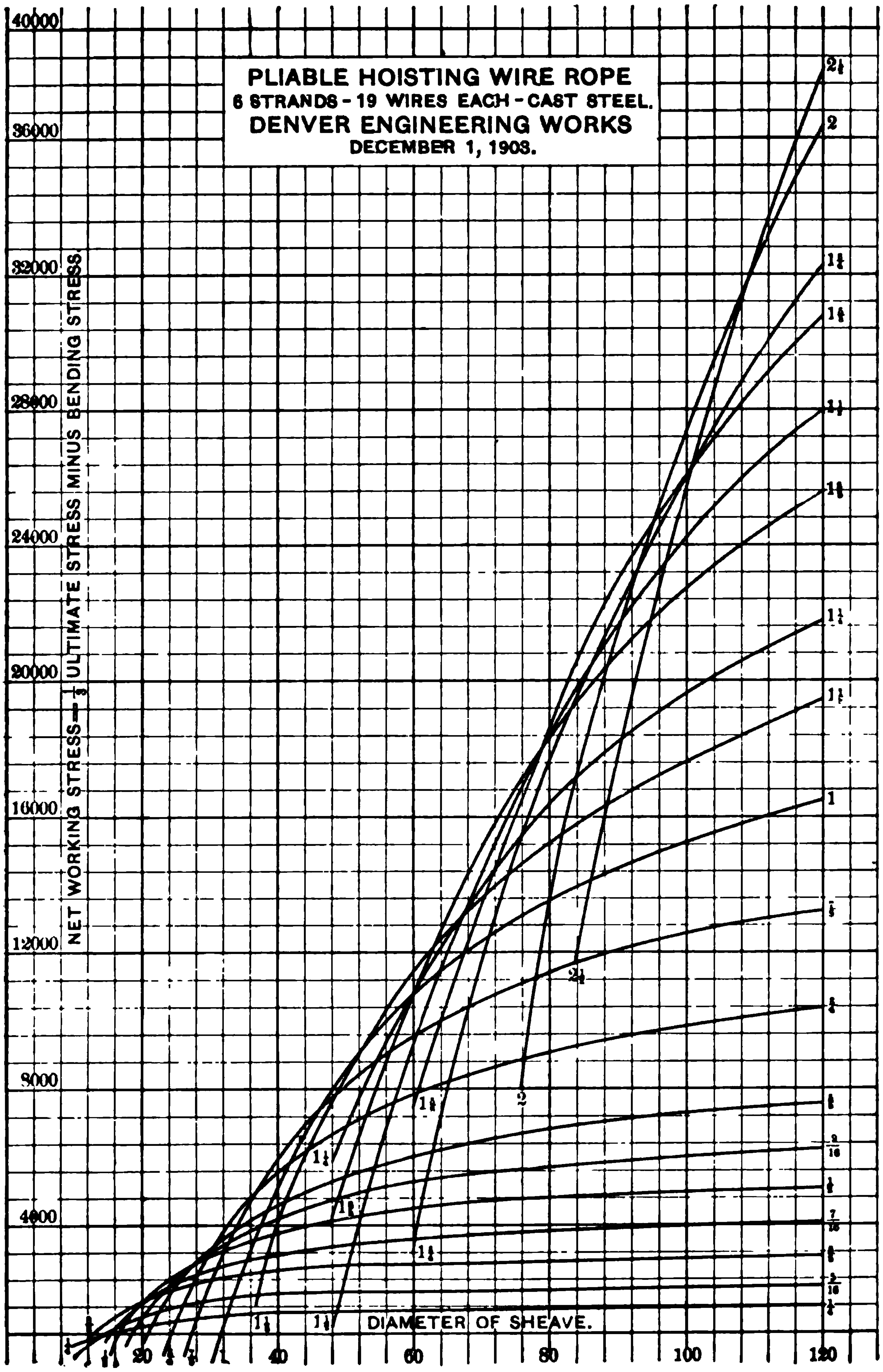


FIG. 76.—The Net Working Load of Ropes over Sheaves.

one-seventh or one-sixth, respectively, of the ultimate strength of the rope. A larger factor of safety would unnecessarily increase the size of the rope, which in turn increases the dead load, due to its own weight and the diameter of the sheaves over which it must pass.

The stresses to which a rope is subjected, in addition to the live load to be carried, are those due to its own weight and friction, its bending over the sheave, its rigidity, and the shock induced at the time of starting. Some of these can be calculated, but the tension induced by the rigidity of the rope is not so easily ascertained. An initial strain is set up in the wires by the bending around the drum or sheave which is not equally distributed throughout the rope. Its value depends upon the sharpness of the bend and is ascertained by the formula given below. A small sheave and a large angle of deflection produce a large bending stress.

The Resistance Due to Bending.—This must be determined before the available load which the rope can carry can be ascertained. Usually the live load represents the difference between the safe or minimum allowable tensile stress and the stress due to bending. In deep shafts regard must also be taken for the weight of the rope, which is to be subtracted from the working load.

The bending stresses, k, which are developed in a rope when bending over a sheave or hoisting-drum through an angle exceeding 30° are indicated below.

Let X = the diameter of the steel rope in inches;
r = the radius of the sheave or drum in inches;
t = the thickness of the flat wire rope in inches;
k = the bending stress in pounds.

Then for a 7-wire rope

$$k = \frac{730{,}350X^2}{r+0.3X};$$

for a 19-wire rope

$$k = \frac{367{,}000X^2}{r+0.5X};$$

for a flat wire rope

$$k = \frac{89{,}465t}{r + 2.23t}.$$

The diagram in Fig. 76 graphically shows the net working load which a given rope of pliable steel can carry over a pulley of a given diameter. This diagram is based on both formulæ, and is taken from a treatise of the Trenton Iron Co. on Wire-rope Transmission. As an illustration of its use let it be required to determine the net load for a 1-inch rope on a 100-inch sheave. At the intersection of the vertical line from the point on the top line marked 100 inches to the curve representing the 1-inch rope, the horizontal line carried to the right-hand edge shows 15,500 lbs. as the net load. It will be noticed that a 2-inch rope on a 76-inch sheave has a net working load not greater than that of a $\frac{3}{4}$-inch rope on the same sheave. This diagram, therefore, indicates the reduction in the capacity of the rope due to bending stress.

Elastic Connections between Rope and Cage.—Usually while lowering to a platform an excess of rope is paid out and tends to coil upon the top of the cage. This operates disastrously upon the wires of the rope, which become bent and ultimately broken. A flexible connection, therefore, is provided of a length sufficient to allow for the probable slack. This consists of a short length of chain with a snap-hook (Fig. 77), which can be released when desired. Owing to the shock which these links receive, they must be carefully examined at short intervals. In some mines it is the practice also to anneal the iron by heating to a red heat and cooling slowly. This is done once a month.

FIG. 77.

These connections should be as short as practicable in order to diminish the amount of shock induced at starting. Carelessness in paying out excessive slack reduces the life of the rope and increases the stress at starting.

Results of observations in Prussian mining districts during

the past twenty years showed the effect due to repeated shocks at starting to be more disastrous to the flat than to the round steel ropes. In certain districts 6 per cent of flat steel ropes broke suddenly out of 1419 in use; 12 per cent of flat charcoal-iron ropes gave way suddenly; 7 per cent out of 124 flat Manila ropes broke without previous warning; 2 per cent of 5527 round steel ropes gave way, and 10 per cent of 1350 round charcoal ropes failed without giving previous warning.

The tension due to jerks is twice that due to the dead load with 6 inches of slack, and more if the slack is greater.

Let T = the total tension at starting;
a = acceleration at beginning of hoist;
h = slack in inches;
b = elongation of rope due to total load, W.
W = weight of car, cage, mineral, and rope.

$$T = W\left(1 + 0.5\sqrt{1 + \frac{ah}{16.1b}}\right).$$

Rope Sockets.—There are two types of sockets for round ropes—the conical and the double-pin, the former being the stronger. The conical socket (Fig. 78) is slipped on to the rope, the wires are untwisted, hemp centres cut out, the wires bent back and forth into a tangled mat to fill, as nearly as possible, the conical socket, which is then slipped into place. This is slightly heated, and soft lead poured in to solidify the mass. The socket and rope are surrounded with wet clay to prevent heating of the wires beyond. The double-pin is treated in the same manner, but its connection with the chain is by a pair of pins through the links, instead of a ring for hooking, as in the former case. A "goose-neck" socket consists of a pair of trough-shaped tongs, bent to a loop, and riveted to the rope by three or four rivets, driven cold. Flat ropes have riveted to them shackles with eyes, which receive the first link of the chain. Six inches of the end are untwisted and doubled back, bound with wire, the shackles slipped on, riveted

FIG. 78.

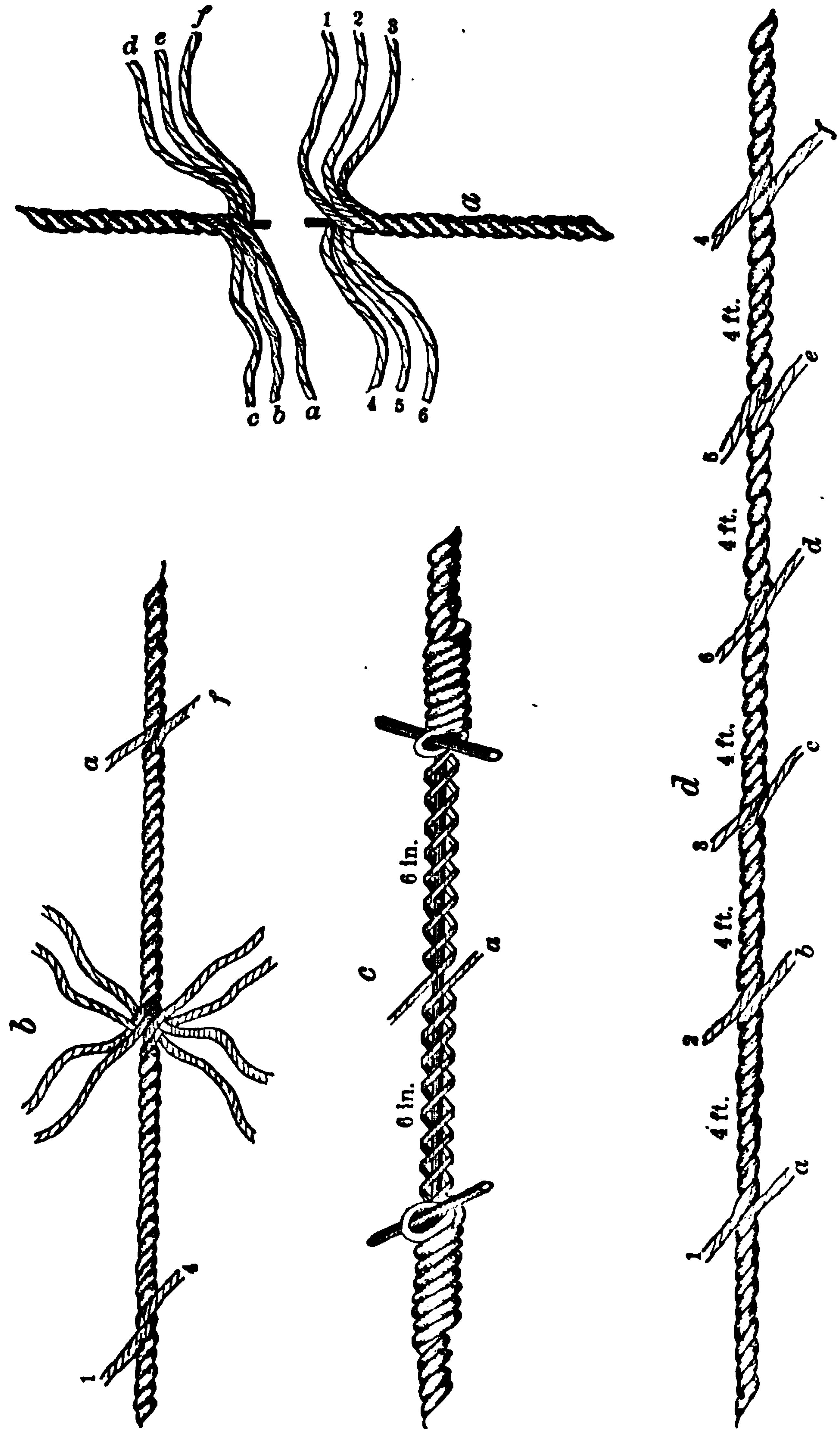


FIG. 79.—The Method of Splicing Ropes.

through the rope, and the hoops finally slipped on and driven tight.

In securing the rope on the drum it is only necessary to continue several extra coils of the rope, insert the end through the wooden lagging, and fasten it on the hub or shaft; or, instead, the end may be bolted to the arm of the casting. If the fastening have but a 10-lb. grip on the rope, it will resist a weight of 90 lbs. if there is only one coil around the drum; if there are two extra coils, 800 lbs. will not budge the 10-lb. grip; with three extra coils it requires 7300 lbs.; while with four it has a 65,000-lb. resistance and can support that amount of tension.

Wire rope is spliced in the same manner as hemp. The strands are unlaid for 3 feet, and each passed over one and under another of its corresponding strands on the opposite rope, for a like distance; the free ends are then trimmed off close. Short-twisted rope is more easily spliced than one of long twist.

The Head-frame is designed to withstand the stresses from the weight of the dead and live loads on the rope and their frictional resistance operating vertically, and from a pull of the engine along the inclined rope. The latter exceeds the former by an amount equal to the frictional resistance at the sheave and its bending stress. The frictional resistance may be taken as one per cent of the gross load on the journals, and the bending stresses of the rope may be determined from the preceding formulæ. The combination of these two forces produces a resultant operating in a direction which nearly bisects the angle between the two ropes. From these, with the load, may be obtained the direction of the resultant. Its line should fall nearly central within the base of the head-frame, when the latter is of wood. It is not so essential if the derrick is of steel.

The frame over the shaft is built in one of two patterns. The form of the derrick is essentially two vertical right-angle triangles, each upright and brace being respectively parallel to the two directions of the rope. The apex of the triangle should be at the centre of the axis of the sheave. The triangular frames are connected in a transverse direction by braces and ties. Instead

of single sticks for the uprights, a vertical frame is built of four uprights surrounding the shaft-opening with two or three struts projecting from their top toward the hoisting-engine and inclined to furnish ample base for the derrick.

In the other pattern the four posts are batired outward to afford an ample square base of support without the use of any additional struts (Fig. 80). The frames may be built of wood or steel, the latter being extensively employed in collieries where the speed of hoist is great (Fig. 81). The steel frames are built by construction companies to order and are composed of the usual structural shapes in steel.

The foundation may be either masonry pillars or concrete, extending some distance outside of the lines of the shaft. The lower base consists of three longitudinal sills framed with three cross-sills. Bolted to the former are cast-iron shoes to receive the uprights and braces. The base-frame is bolted and anchored to heavy timbers buried in the ground or in the concrete.

The Height of the Derrick.—The essential requirements of the head-frame are stability and safety. The former is obtained by a broad base and a rigid construction of frame; the latter is obtained by its construction of fire-proof metal and a height sufficient to satisfy the conditions of hoisting. The risk of fire, the effect of weather, the working of the joints, and the difficulty of securing sound sticks of requisite length render the adoption of material other than wood advisable. Height, which is an essential feature of the head-frame, is difficult to attain without a rigid frame, and this implies steel construction. This is the rule at large mines. Steel frames are more desirable and immeasurably more rigid than wooden ones. To withstand injury from the continual vibration, pin connections are more suitable than riveted connections.

The height of the derrick should be such as will furnish security against overwinding by providing sufficient margin within which the engineer may stop the hoist. The greater the speed of the engine the greater is the necessity for a high derrick. The minimum allowance is that which corresponds to the length of

two coils of rope on the drum plus the height of the cage. The distance from the landing station for cars to the sheave would give the engineer four or five seconds in which to secure control of his engine. Usually the derrick is of a height twice the amount stated. In anthracite regions it is often more than this.

Steel frames in some instances are 65 feet in height, the base of the first-mentioned type being about 16 feet square with the sides extended for the braces. Of the second type there are numerous steel frames with base of 25 to 30 feet square, with a height of 50 feet. Derricks are frequently extended to a height above that of the tipple or breaker platform to permit of a continual descent of the mineral from the time of unloading to the time of shipment. In such cases as these the head-frame is over 100 feet in height. Those in the zinc-mining districts of Missouri are of the four-post type, 30 feet high, with suitable braces. Some in the anthracite district are as high as 130 feet.

The hoist-frame is rarely enclosed, and preferably is not housed because of the danger from fire. A fire breaking out in the building over a shaft would be uncontrollable with a draft from below, besides risking the communication of the fire to timbers of the shaft. A fire communicated to the hoist-frame from below would endanger the only means of exit which the men have from underground.

The tipple type of head-frame is built of bents, suitable for the screens and pockets. It connects with and extends from the head-frame in a direction opposite to that of the back-stays. It is of wood or steel with all members carefully calculated. The rhythmic motion set up by the vibrating screens must be provided for, and the design varies radically between one having a dump in the plane of the hoist-rope and one in which the shocks of dumping are at right angles to that plane.

Slopes require no high head-frame, for the line of track extends directly from the shaft mouth to the elevated structure, and in reality forms the back-stay for resisting the overturning tendency of the structure. This is the strongest possible type of frame, for the resultant falls inside of the base of the tipple.

FIG. 80.—A Timber Head-frame.

The Sheave.—Upon the top of the uprights is a frame which supports the sheave on a short horizontal axle. It may be built directly upon the braces and uprights of the head-frame or upon the platform above it. The dimensions of the sheave are determined by the size of the rope. It consists essentially of a spider frame carrying a grooved circumference filled with some material

FIG. 81.—A Steel Head-frame at a Bituminous Colliery.

which has a large coefficient of friction, as, for instance, wooden blocks on the end, or rubber. These latter prevent slip and render the hoisting more secure, as well as protect the rope from excessive wear. Its hubs are double, connected to the rim by rods let into sockets. The entire construction is made as light as possible consistent with strength, to reduce the inertia and avoid the abrasion that occurs with a heavy wheel continuing to revolve after hoisting has ceased.

The cost of the high steel head-frames of a mine averages $32.20 per ton of material, including the design and details. The erection averages $13.00 per ton when the connections are riveted.

As regards the cost of wooden head-frames, the following may afford a suggestion for average conditions. The four-post frame for small mines of 31 feet requires 1800 feet of board measure and costs from $100 to $150 for labor and supplies. Larger frames without back-stays for the height of 50 feet consume 14,000 feet of lumber and cost $550. Tipple head-frames for vertical shafts, with 60 feet of height, require 15,000 feet of board measure and cost about $800. Slope tipples for inclined shafts, built to a height of 31 feet, are completed for $3300, using 40,000 feet of board measure.

Cage Indicators.—Several independent devices are used to indicate the position of the cage in the shaft at all times during its hoist. Automatic indicators are installed in view of the hoisting engineer. An indicator has a pointer which moves around a dial or vertically along the side of a column, the dial or the column being graduated to the depths of the shaft- or slope-landings. A string winding on a miniature counterpart of the hoisting-drum moves the index a distance commensurate with that of the cage and shows the latter's position after starting and its proximity to the landing station during the hoist. The approach of the cage is also indicated by an electric communication automatically established by the cage in its ascent. In this arrangement the cage makes an electric connection as it passes a given point in the shaft below the point of landing, which is signalled to the engineer. A second notice is given in a similar manner when the cage is just about to reach the landing. This notification is independent of the automatic devices for the guidance of the engineer in braking the engine. In addition to these means, the brakeman or landing-man at the head of the shaft having charge of the cars may signal the engineer as the cage approaches the top and furnish another control, which, being sentient, is always to be preferred to any mechanical device, no matter how perfect.

Signalling.—The communication between the employees below and the engineer above may be had by the use of a wire with a bell or some electric annunciator, or by a speaking-tube or telephone. Whatever may be the means employed, it is essential that it should be quick and accurate, presenting no opportunity for mistakes. The early clumsy arrangements depending upon the signal of the gong, bell, or triangle which is struck by a lever, operated from below by a rope or wire, are untrustworthy. The pull on the rope may be too light to strike the blow, or more strokes may be rung up than are intended, and there is no means of making a correction before the engineer will have proceeded to answer the call. Accident may then ensue. The annunciator is another method which is quick and convenient for signalling to the engineer or between stations, but its scope is limited to a very small set of signals. Unquestionably electricity offers the best means of signalling between stations in dry shafts. One wire or a couple of wires may be placed within easy reach of those in the cage, furnishing a means for its occupants to signal to the surface or elsewhere while in motion. Such wires would also be convenient for those engaged in examining or repairing the shaft, being connected to electric gongs located at each landing-station.

A uniform code of signals among mining men is an eminently desirable feature. A hoisting engineer at a new mine may err with serious results by an interpretation of a signal which elsewhere represents some other operation.

The Telephone.—There need be no argument to convince the reader of the advantages of the speaking-tube or telephone. This safe natural means is largely employed. It is rapidly supplanting the electro-mechanical gongs. The expense of laying out a mine telephone system is generally much less than that of a surface system, since in most cases the wires can be supported by means of porcelain or glass insulators, either from the roof or from any timber which is used on the sides of the road. Their greater expense is overbalanced by the great saving in time, and their efficiency in case of accident. For

deep shafts, telephones are almost imperative; in fact, mining legislation in some States compels their use.

Bridging telephones, Fig. 82, connected to the metallic signal-line, used in connection with rope haulage, reduce the cost of installation to that of first cost of the telephones and the nominal cost of connecting up. For insulation, rubber-covered wire

FIG. 82.—A Mine Telephone.

will serve generally in localities where moisture or contact with timber might cause an interruption of service. When an installation of mine telephones will warrant the extra cost, lead-covered cables should be employed.

Means of Increasing the Safety of Ascent and Descent into the Mines.—The necessity for the introduction of safety appliances of this kind needs no argument when one considers that the engineer, from a point somewhat remote from the elevator, is raising the latter, at an hourly rate of speed reaching 30 miles, and the load, whose weight may be nearly one fourth of the strength of the rope. This is to be landed promptly, accurately, and without shock at the precise point. In this work, which is repeated scores of times each hour and with an engine of several hundred horse-power, with no guide except the index moving about a small circle, any defect in the operation of the indicator, or any momentary distraction of the engineer's attention, may result disastrously and the engine raise the cage a few feet beyond the point intended, and indeed may raise it over the sheave at the top of the frame. Again, some defect in the operation of the throttle or of the brake may prevent the engineer from bringing the engine

to rest as promptly as desired, and one revolution or two, which may wind 30 to 50 feet of rope, may bring the cage to the danger limit. Recalling that these operations are performed 50,000 times per year, the mental strain and the anxiety of the engineer where no overwinding device is provided is unquestionably serious. Again, coal and men are hauled on the same vehicle; and if no proper method of signalling exists, the engineer may hoist the latter at the speed and to the place of dumping the former.

Though the number of accidents arising from overwinding is small, nevertheless it is sufficient in amount to warrant the introduction of some device. There are two general classes of these. The first type is an ample detaching-hook, and the second an automatic control for the engine.

Detaching-hooks.—Safety contrivances are installed at the head-frame for preventing a cage from falling when severed from the rope through overwinding. Their design contemplates the release of a link between the rope and its cage, and the simultaneous action of some support which takes hold of a portion of the framework of the headgear. These give effective service and are employed very extensively in Europe, but to a lesser extent in America.

Ormerod's hook is one of the best of these safety links. The apparatus when in ordinary use, as in 2, Fig. 84, is wider at the bottom than at the top. When overwinding occurs the link is drawn into the bell-mouthed cylinder just below the sheave. This lower part comes into contact with the cylinder, thereby closing the bottom part of he link and expanding the top part. Its projections catch over the top of the cylinder, while at the same time the rope shackle *A* is forced out of its seat, thus being allowed to go free; the bottom shackle, *B*, drops into a slot and locks the link firmly in its position. The cage, being suspended from the chain, cannot fall back. To prevent the possibility of the link becoming disarranged in ordinary work, a small pin is inserted centrally through the plates, which pin is sheared off as the apparatus passes into the cylinder.

Middleton's hook, 3, Fig. 84, operates on the same principle.

King's hook, one form of detaching-hook, is illustrated in Fig. 83. It consists of two outside plates, enclosing two inner ones, capable of oscillating about a strong pin which passes through the series. When hoisting has reached the danger-

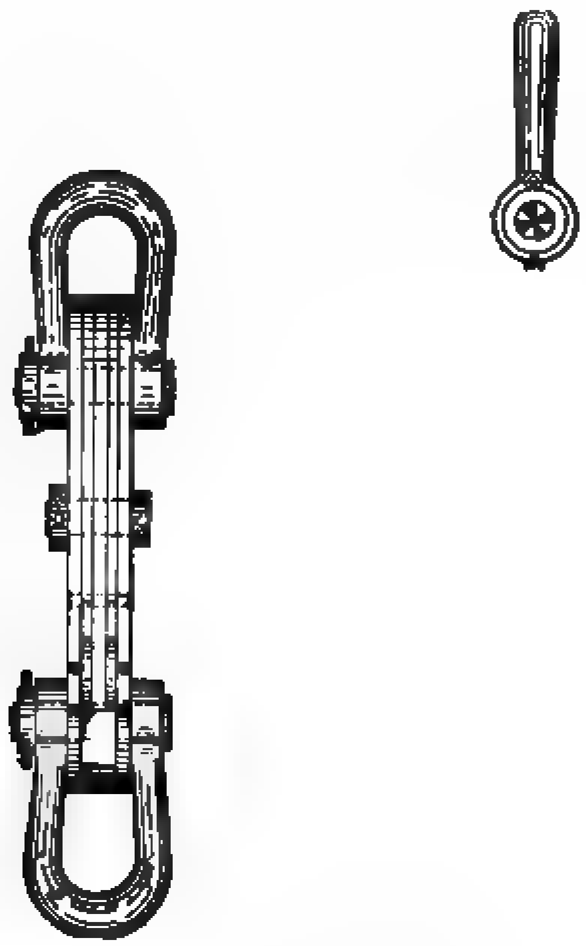


FIG. 83.—King's Detaching-hook.

point, the wings on the sides of the latter pair strike the plates, the jaws permitting the rope to be hoisted over the sheave and the cage to fall. At the same time, however, the projecting wings on the hook catch on the protruding edges of the platform and hold the cage.

Walker's safety attaching-hook, 4, Fig. 84, has a loop which encircles the hook and is bound by it to two copper rivets. These are sheaved when the hook is down. The jaws then open and release the rope, locking the suspension jaws on the disengaging-plate and holding the cage.

Engine Controllers.—This class is a more or less flexible connection between the shaft and the engine by which the cage

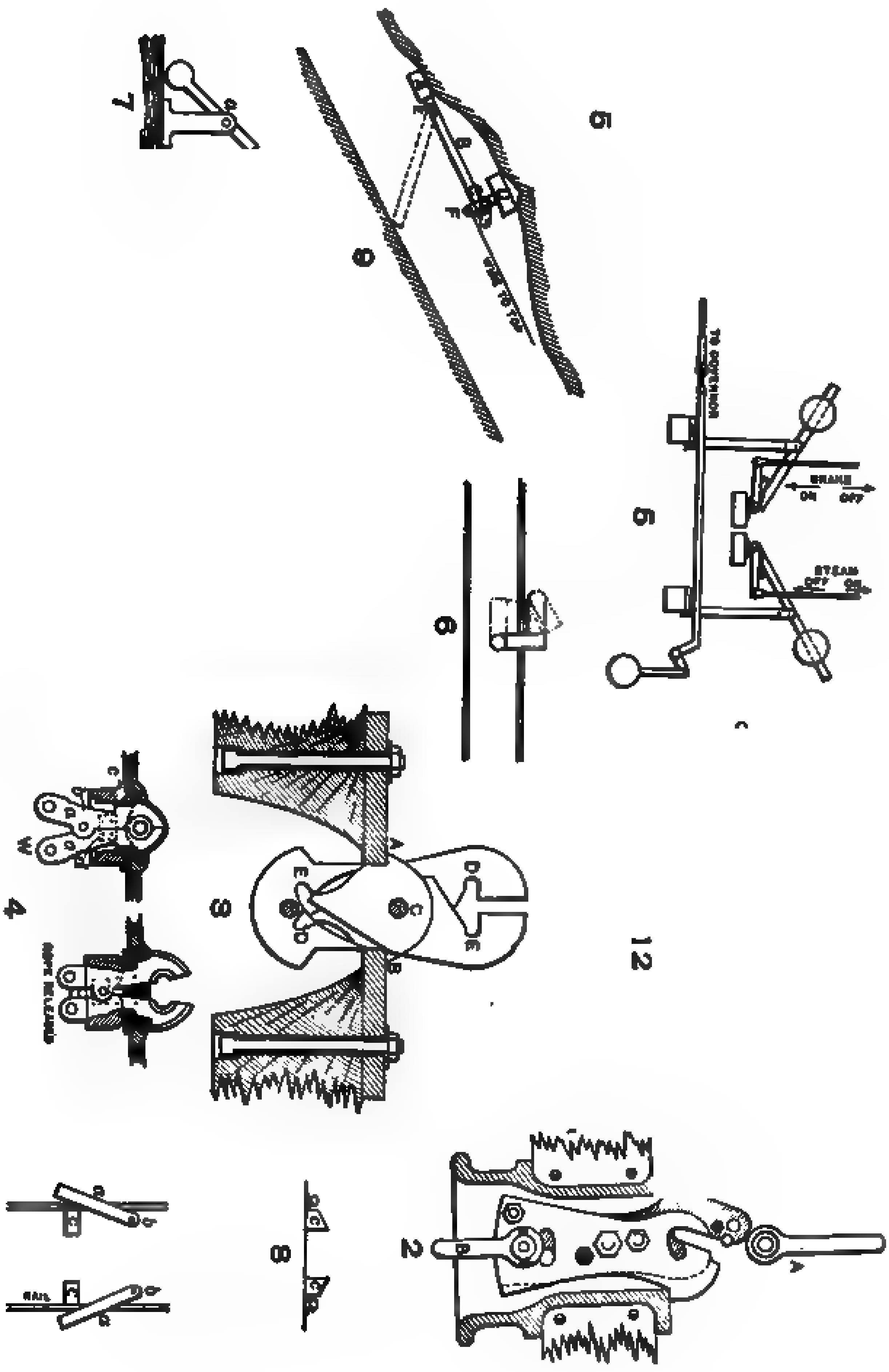


Fig. 84.—Overwinding and other Safety Devices.

automatically shuts off the supply of steam or applies the brake when it reaches a certain point.

The Visor, 5, Fig. 84, consists of a governor suitably driven by a crank-shaft, which, if the speed of the engine exceeds a certain point at the end of the hoist, applies the foot-brake through a combination of levers, and closes the steam-valve.

Buckets—Kibbles.—The simplest conveniences for mineral hoist are employed during sinking and during the mining operations of small mines. These are sheet-steel tubs, or buckets, fitted at the top with a bale of round iron and at the bottom with a small ring. The bale is hooked into eyes opening into a strap which is bolted to the sides and under the bottom of the tubs. The snap-hook (Fig. 85) on the hoisting-rope catches into the ring of the bale. When, however, buckets are to be frequently

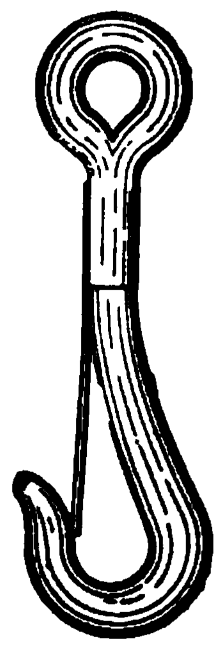


FIG. 85.—Snap-hook.

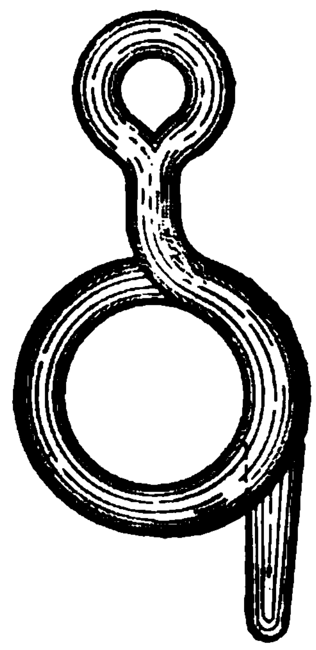


FIG. 86.—Serpentine Hook.

detached from the rope, serpentine hooks (Fig. 86) are used instead. They are plain cylinders of sheet steel or boiler-iron, 18 to 33 inches in diameter, 30 to 54 inches deep, having a capacity of 600 to 3000 lbs., and a weight of about one quarter of their contents. Frequently they are bellied and used in slopes on skids. The buckets are loaded from chutes at the mill-holes in the mine (Fig. 87), and transported on the trucks to the shaft and raised to the surface.

The dumping of the bucket at the surface may be accomplished in one of two ways. It may be lowered upon a truck and removed from the hoist-rope and taken to the point of empty-

ing, meanwhile being replaced by an empty bucket which is lowered below. On the other hand, the bucket may be emptied at the surface without removal from the rope. In the becket and high under the bucket the shaftman inserts the hook of a short length of stationary chain. The tub with its hoist-rope is

FIG. 87.—Loading-gate for Buckets and Cars.

then lowered and inverted, its contents being emptied upon a grating at the side of the shaft or into a car which is run over the landing-doors which have been lowered across the shaft. In Fig. 89 is another device, not so safe, in which the bale is pivoted a little below the centre of gravity of the bucket, and is held in position by a loose ring on the bale, slipping over the pin at the upper rim of the bucket. To dump, the brakeman merely raises the ring and allows the bucket to reverse. It is easily righted and again fastened.

Aside from objection to hoisting of buckets on account of its low speed and insecurity, there is a more important one in metal-mines, namely, the production of considerable powdered material. This soft product, though it may reach the surface, probably never reaches the smelter and, whatever its value, is lost.

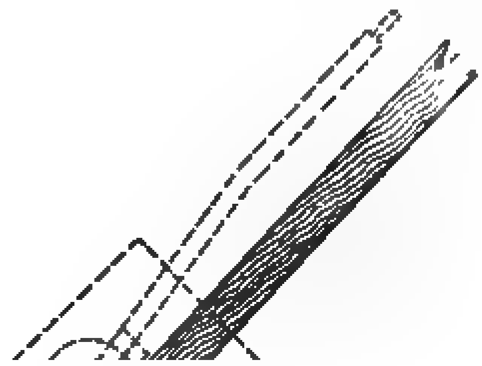
As the softer minerals are also the richer, one can readily see the necessity for avoiding too much handling of the mineral. In addition to this is the risk of collisions, unless each tub has

FIGS. 88 and 89.—Mine Buckets.

its own hoist compartment with smooth-lined guides. These slides permit of more rapid hoisting, but the damage and accidents arising from contact with buckets when the end of a plank protrudes make the risk and cost quite considerable.

Shaft-guides. — To facilitate rapid hoisting, the shafts are fitted with conductors or guides, along which the conveyances move. With a bucket-hoist on the slopes the floor is planked with 4-inch wooden guides spiked on either side of a runway, between which the bucket may slide in its passage. The ends are finished so that they will not easily loosen and cause accident. Experiments with steel rails for guides have proved them far too noisy for comfort.

When a bucket is hoisted in a shaft it swings freely, but is guided by two or four ropes stretched from top to bottom; these pass through holes at the ends of two or four arms which make a horizontal frame. This frame rests freely above the shackles of



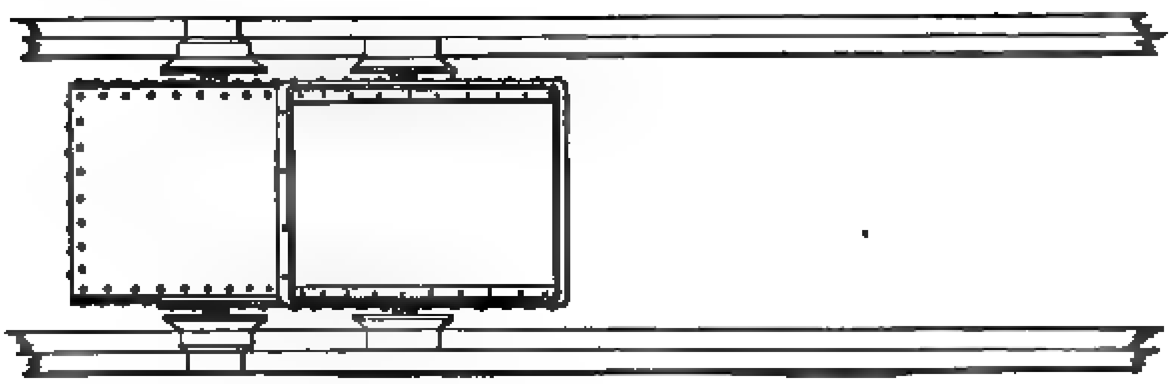


FIG. 90.

the hoist-rope and keeps the bucket in a central position during all times of the hoist. Cages are guided by two 4-inch timber strips which are bolted to the buntons, or cross-pieces, fixed across the pit. These are securely wedged against the rock, or

are framed into the timbering of the shaft. One guide on each of the two sides is sufficient for a cage compartment.

Skips, or Gunboats, are commonly seen in the inclined shafts

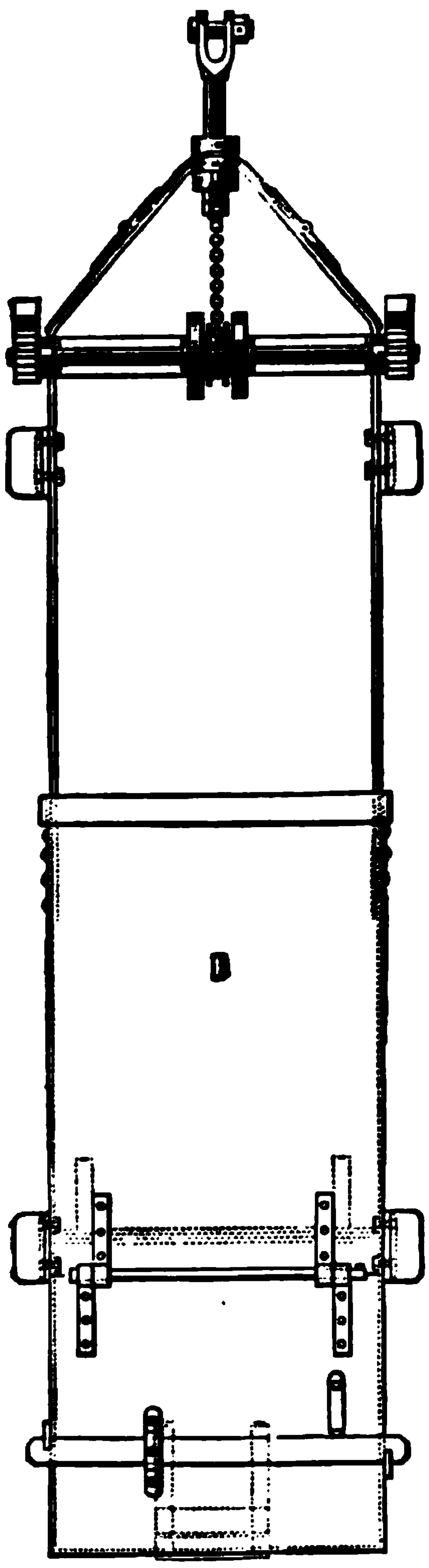


FIG. 91.

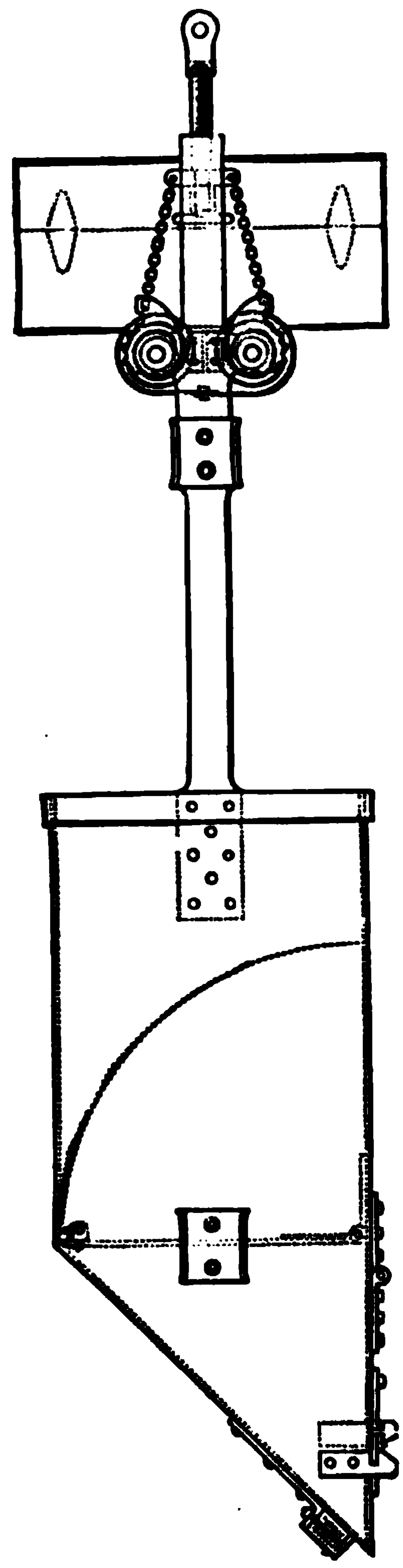


FIG. 92.

for hoisting purposes. They consist of an iron box weighing from 900 to 1500 lbs. set on four wheels held by bolts riveted

at the sides; the front view is rectangular, but the side view is trapezoidal (Fig. 90). The inclined end is uppermost with automatic dumpers, while in those discharging by door it is below. The hoist-rope is attached to the bale, which rotates on a pin passing through the side of the skip, often back of the centre of gravity, so as to dump automatically. The charge of one or two car-loads is shovelled or chuted into the skip, which empties at the surface into a bin or on grizzly. In one variety the contents are discharged by the mouth at which it is loaded, while the other form has a swinging door opening at the upper side. A vertical safety skip is shown in Figs. 91 and 92.

The automatic dump is simple; the rear or lower wheels are of wider face than the fore wheels. As the dumping-plat is reached, the guide-rails on which the wheels have been travelling gradually bend to horizontality, and these the front wheels follow. As the hoisting continues, the wider rear wheels catch and roll on a pair of outer guides, and continue up the slope. By this means the lower end is elevated and the skip emptied.

The brake is generally a drag, consisting of a bar about 4 feet long, trailing on the floor, and only catching if the skip breaks away on its up-trip. Often the wheels are confined between two guides on each side, as, for instance, where the direction of the slope changes. The sole objection to these skips is the double and treble handling involved. The car from the mill-hole empties into the chute, whence the skip is loaded, and at the surface the reverse operation takes place.

The Slope-carriage dispenses with two handlings, taking the car at once to the surface. This is simply a double triangular frame, large enough to accommodate a car with two rails on its horizontal top, and two wheels on each hypothenuse. A hook or lock holds the car while riding. For convenience the loading and unloading gangways are not on the same level, the track for "empties" being 6 feet higher than the "full" track. If the seam is too thin to allow of this, each track has a curved roadway connection with the gangway.

The head-room required for the slope having this means of

haulage implies large area and extensive timbering, particularly as a double trackway is necessary if the output is considerable. There is not much necessity for a carriage, except in slopes over 60°.

The Cage.—The cage is essentially a wrought-iron or steel platform for the cars traversing the pit. It is also the usual

Fig. 93.—A Safety Cage with Keeps.

means of transport for the workmen and all other materials between the surface and the different landing-stages in the shaft. The pit timber, workmen, tools, fodder and water for the horses,

and frequently the horses themselves, are lowered by means of the cage. Its form is governed by the shape of the compartment of the shaft in which it runs, and its size by the number of cars. Some cages are fitted with two platforms or decks. On each deck are rails for one or two cars, and so placed as to allow of those on the cage being displaced by others to be hoisted or lowered. The platform is held on the sides next to the guides by two stout iron frames united at the top by a cross-bar to which the hoist-rope is attached (Fig. 93). Two iron ears at the top and bottom on each side confine the cage to the guides. An iron roof or bonnet over the cross-bar shields the passengers from falling rocks.

Some cages are also provided with extra covers, either fixed or adjustable, above the tops of the cages, to protect the men while examining or repairing the shaft.

An automatic device controlling the delivery of the cars in and out of the cage is illustrated in 11, Fig. 84. It serves for the intermittent delivery of cars without an attendant.

Securing the Cars on the Cage.—There are various modes of keeping the car secure in the cage during its ascent or descent. One of these is a "false bottom" in the cage, which sinks an inch or two below the other and outer portions of the deck, but is lifted when the cage is at a landing-station. The deck floor is then on one level, allowing the tub to be changed.

Another device is a bar running across the cage, and at either end is placed a short lever, which turns down or up on being pushed; when down it covers the ends of the tubs and prevents their moving; when up it allows them to be changed. Sometimes a latch-lock on the floor secures the cars in place.

Multiple-deck Cages.—Single-deck cages are almost exclusively used in America, and are sufficient except for narrow, deep shafts, when heavier loads are necessary for large output. Then cages with two or more platforms are used. The landing-stages are arranged in the same number of tiers, from and upon which cars are simultaneously run, without moving the cage. With ample facilities for "decking" the cars, the saving in the trip-time, per car, increases the capacity of the plant. The necessarily

complicated underground stations are an objection to multiple-deck cages. The double-platform carcage is relatively lighter than the single-deck two-car cage.

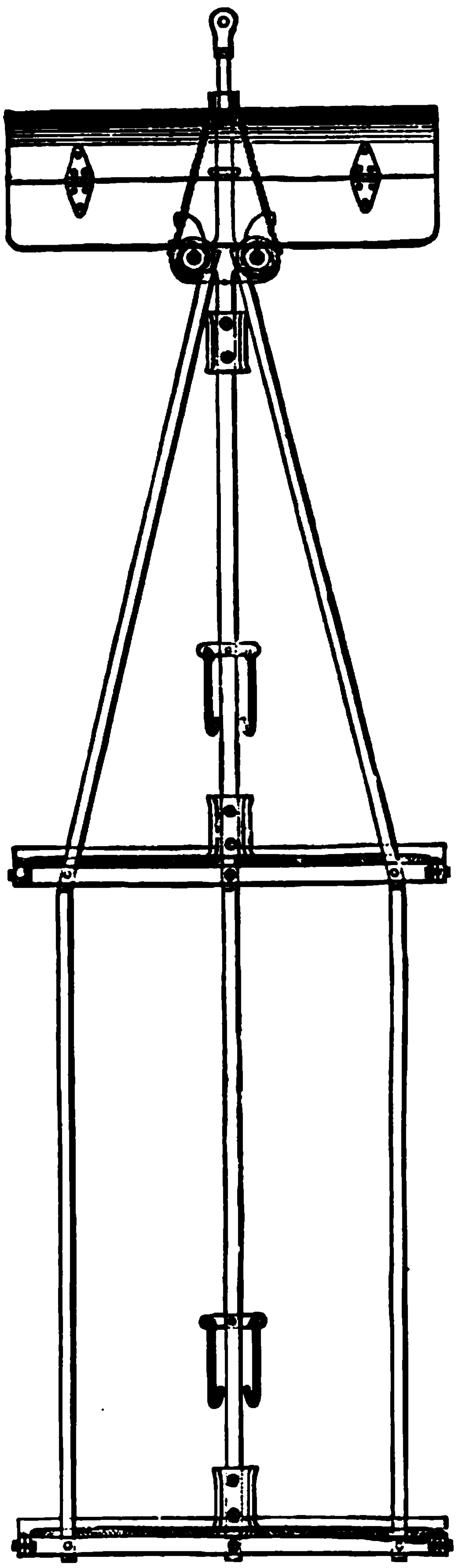


FIG. 94.—A Double-deck Cage.

For inclines of uniform slope, as well as for vertical shafts, cages may be had, the platform being hung on an adjustable lever; but the **slope carriage** would be better.

Safety Clutches.—Various contrivances are proposed for guarding against accidents resulting from the breaking of the rope. The sudden starting and stopping of the hoist produces a shock which strains the cable more severely than does the mere weight. Without care in hoisting and rigid timbering the rope cannot be insured against the jerks. To avoid the accident sure to follow, some form of safety appliance must be thrown into action, to arrest the fall of the cage or skip. The safety catches differ in design and efficiency, but depend generally upon a spring, so held between the rope and the cage as to be compressed, while the weight of the cage strains the rope, but acts on a clutch that grasps the guides and stops the fall, if relaxed by rupture of the rope or otherwise. The clutch is either a pair of sharp-pointed steel levers, which are thrust outward into the timbers, or a serrated cam, the wider part of which will

be turned against the guides and clutch them on either side (Fig. 93). The heavier the weight the stronger will be the grip after they have once taken hold.

In Callow's attachment, 13, Fig. 84, a heavy weighted lever on top of the cage, held up by a spring only so long as the rope is intact, is set into action and engages the clutches upon the guides when the rope breaks. Many a life has been saved by them, but in many instances also they have failed to operate. A momentary check, or any sudden change in the speed, often unnecessarily throws them into action. On the other hand, they are rarely, if ever, in order when the emergency arises, or the guides are so wet and dirty that the clutches fail to catch, if the momentum of the falling cage is great; besides, they are costly and troublesome. Though useful adjuncts, which the law requires, yet it is not surprising that the distrust of them is strong; they give rise to other and more serious causes of alarm. The tendency following their use is to rely upon them solely and to neglect the rope. Others deem them of no value and treat them accordingly. Fortunately, the rope more often breaks at the moment of starting at the bottom than at any other time, and the point of rupture is where the rope enters the socket. If, however, the rope snaps as it turns the sheave—and this is of common occurrence—there is nothing to prevent the inevitable and frightful calamity that follows—the entire length of the rope falls and crushes cage and contents. A simple appliance might be introduced at the top to grasp the rope.

Landing-doors at Shaft Stations.—The landing of the buckets is effected on a hinged door, of double 2-inch planks, lined on top with iron. This is swung against the far side of the compartment, closes it completely, and standing as it does at 45°, the bucket slides into the drift. When not in use it is hooked upright, closing the mouth of the drift and leaving the hoistway clear. That miners may go from one drift to the opposite, an escapement roadway is provided around the shaft in the rock. At the mouth of the shaft are two heavy doors, opened for or by the rising bucket and closed at all other times. These are convenient, and prevent accidents arising from stones dropping down the

shaft, especially while dumping the bucket. A similar arrangement in slopes—a movable drawbridge, lowered at pleasure—receives the car for attachment to, or after detachment from, the rope, and closes the gangway when the car is above the plat.

Cage-chairs.—At the surface and at every landing-station the cage is supported upon chairs, known also as "keeps," "fans," or "shuts." They are counterbalanced levers (Fig. 93), which project into the shaft, giving way for the ascent of the cage, and immediately resume their position after the cage passes them. The latter rests upon them during loading. To lower the cage an attendant raises them out of the way while the cage is being lifted off.

With double-decked cages the top deck of the cage at the shaft bottom rests on keeps, while the bottom of the other cage stands on similar supports. When the cars have been changed a signal is given, the cages are lifted, while the attendant at the surface pulls them clear of the shaft until the bottom deck has been lowered below them. Releasing the handle thrusts them into the shaft to hold the cage at its top deck.

Those at the shaft bottom are necessarily handled differently. As the loaded cage leaves the shaft bottom the attendant there pulls the handle of the keeps back and secures it, thus preventing them from protruding into the pit. Holding the lever working the keeps, when the lower deck of the cage has passed below their level, he allows them to spring out to the support of the top deck. The car is then changed, while the lower deck of the other cage is being changed at the surface. The keep-handle will not require further attention from the attendant below until the cage has left for the surface, when he secures the handle back in its place.

Stauss's patent keeps, shown in 15, Fig. 84, have been designed to lower the cage into the pit without a previous lift. They are fixed on wooden spring cantilevers, to take the shock of the cage when it is lowered on them. At 16, Fig. 84, is the end view of another style of lowering-keep. These reduce the wear and tear of the winding ropes and engines. The cage can be low-

ered immediately without the lifting jerks, which are so grea at cause of deterioration of ropes and engines. When these keeps are used the winding rope must of course be carefully adjusted in length so that the cage does not fall after drawing back the keeps. The sinking of the cage should not amount to more than the slack of the rope when unloaded. The length of the rope can easily be adjusted by means of cage-adjusting hangers, which are essentially of the turnbuckle type.

Self-dumping Cages.—The self-dumping cage (Fig. 95) is a bucket supported below the cage, whose floor is divided and hung at the sides like two doors. Below it is loaded from cars, and at the surface it is dumped very easily. Unless the coal is

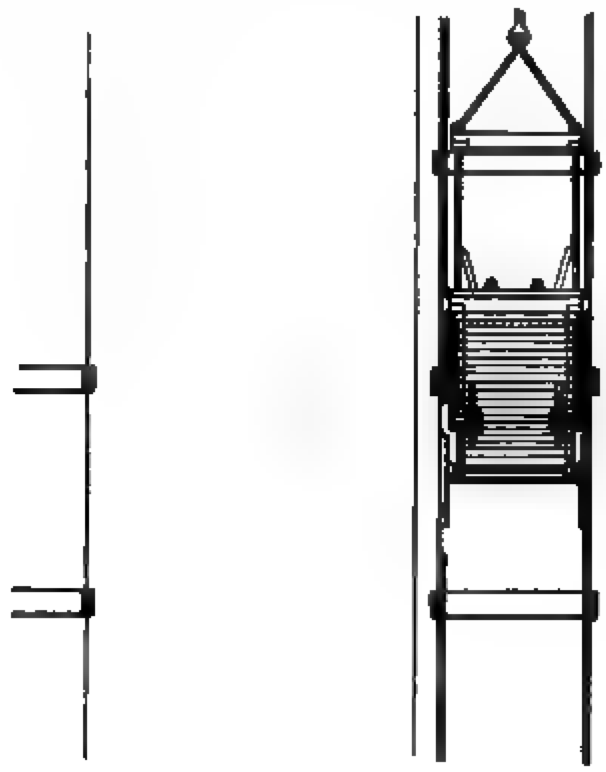


FIG. 95.—A Self-dumping Cage.

to be coked, tender coal cannot be treated in this way. Where all the coal is to be coked, or at a small mine marketing its coal, self-dumping cages are used, only one man being required for the work. The capacity of a pair of them is limited to 1000 tons of run-of-mine coal if screened over one set of bars.

Mine-cars.—The dimensions of cars are reduced to a minimum commensurate with the size of the haulage-way, and they

must be large for purposes of economy. To obtain the highest efficiency they are made as large as the natural conditions of grade in the haulage-ways will allow. Their weight is reduced to a minimum in order to economize on haulage-power. Indeed, the amount of power available determines their maximum allowable gross weight and grades suitable for the haulage-ways.

In small mines where man-power is employed for tramming, the limit of weight and size is fixed by the inclination of the return track; so, too, in buggy-rooms the weight and capacity of the car depends upon the power there employed. These cars are

Fig. 96.—A Metal Mine-car.

therefore, limited to small dimensions. Coal-cars are much larger and heavier when mechanical systems of haulage can be utilized. If the conditions permit automatic delivery of the cars from the face to the locomotive or the haulage-way, their weight is that which the mule or horse can conveniently return empty to the face. Evidently the car should be as light as is compatible with its strength. Its weight is about one half that of its contents. In metal-mines the proportion is somewhat greater. The car weighs from 600 lbs. to 1500 lbs. Some coal-cars carry as much as 120 cu. ft. of mineral, but the capacity of those in metal-mines is less than 30 cu. ft. For strength, compactness,

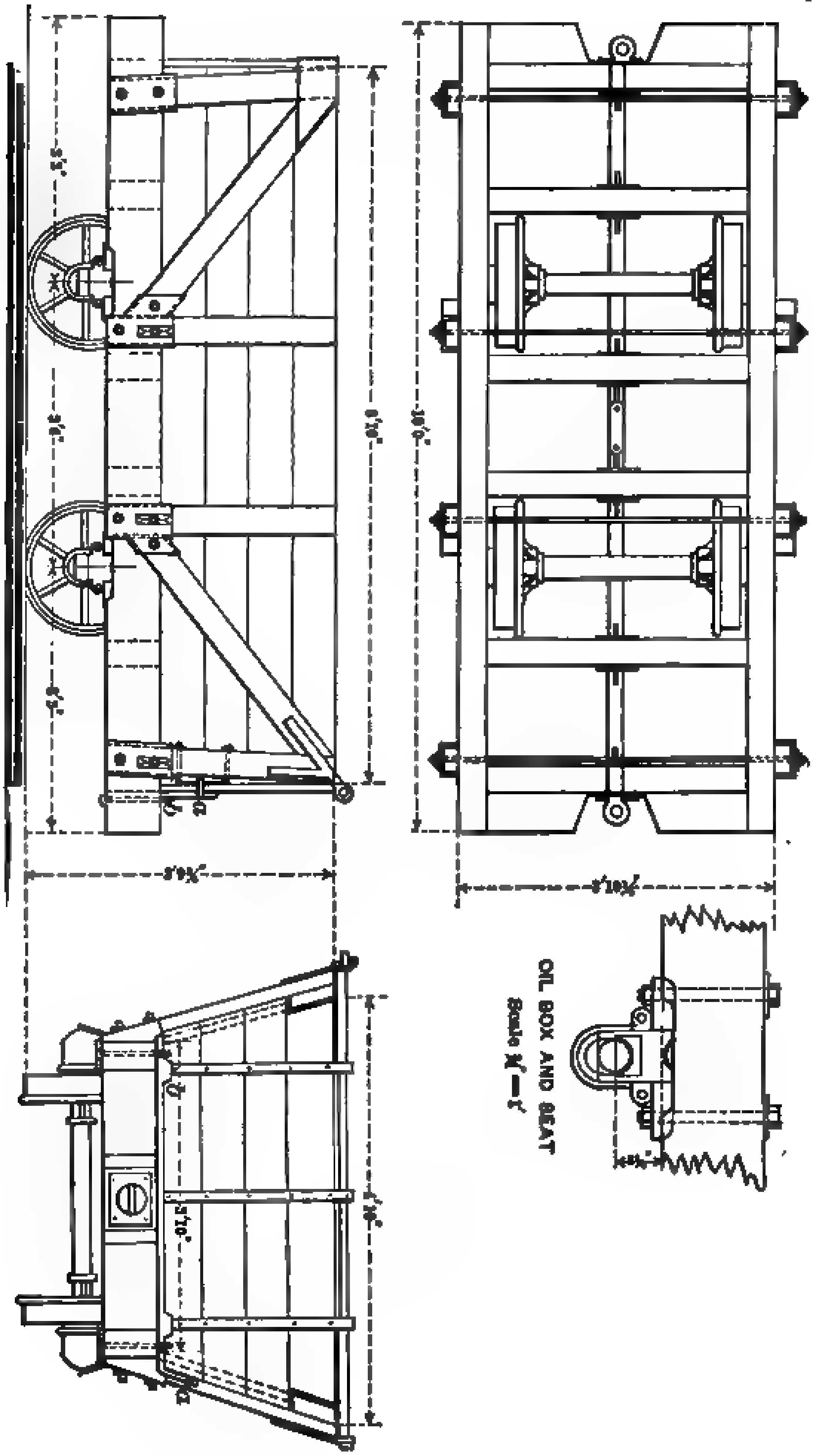


FIG. 97.

and tightness sheet-steel car-bodies are used where the mineral is hard, as in metalliferous mines. Otherwise a wooden box is employed. If the mineral is heavy but soft, it is necessary to employ a sheet-steel boxing in preference to wood

The width of the cars depends upon the gauge and its set. The bodies resting on the axles are preferred, but may not be advantageous because of the wide gauge of track. On the other hand, with a narrow-gauge track and given capacity the car-body is elevated and its centre of gravity too high. A compromise is frequently taken by which a low wide car of great capacity may be raised on a narrow track. In this case the axles are elbowed for large wheels and the car-body is set down on them, but is widened above the wheels to full width of the track. Their width must provide ample clearance for passing men in the haulage, and their length allow for space in the cage compartment. The length is limited by sharpness of the curves. In the collieries the maximum is 9 feet, and in metal-mines it is not over 6 feet.

Their height depends upon the conditions of loading. In metalliferous mines the cars run only in main haulways, and are filled from chutes, provided with a spout and gate, easily manipulated at the bottom; if also hoisted on a cage, their height is a matter of indifference. When the seam is thick and flat and the roof good, they are carried to the face of the work, in which case they are filled by hand. If so, or if raised on carriage, the difficulties and expense are greater with a high car To shovel one ton into a 3-foot car requires over 7300 ft.-lbs.; into a car 4 feet high, 9500. The average man can exert a continuous shovelling effect of 28,100 ft.-lbs. per hour. Allowing for the weight of the shovel, delays, throwing the mineral forward, a shoveller may load about 20 and 14 tons, respectively, in the cars per shift. Even for a medium output the economy is manifest. In metalliferous mines a low car is used, but in collieries heights of 4 feet 9 inches and over are common. For stability, too, a low car is desirable.

Car Details.—The steel cars are purchased of the manufacturers and may be had to order of any size and design for the

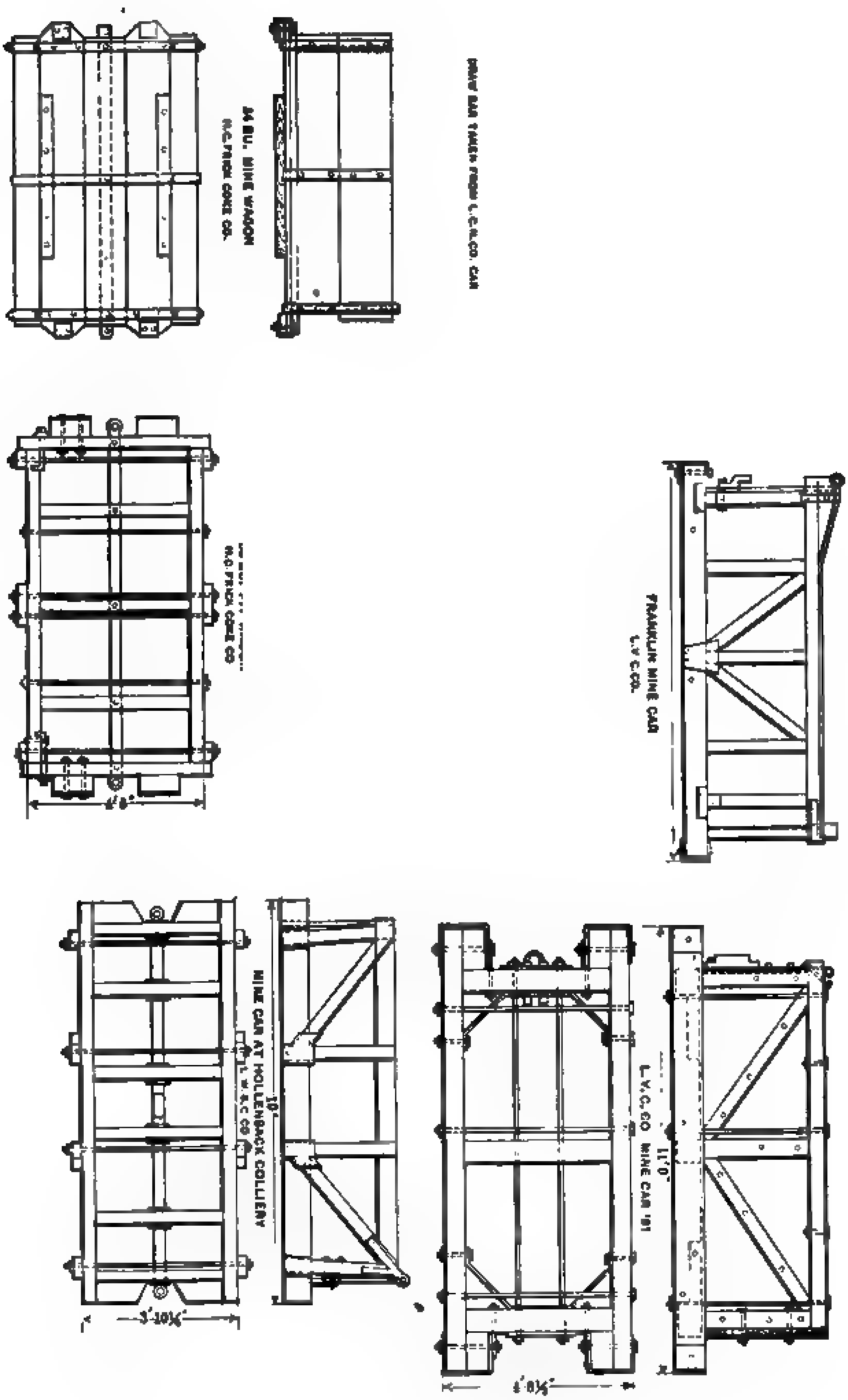


FIG. 98.—Wagon-boxes

POSTS AND PEDESTALS.

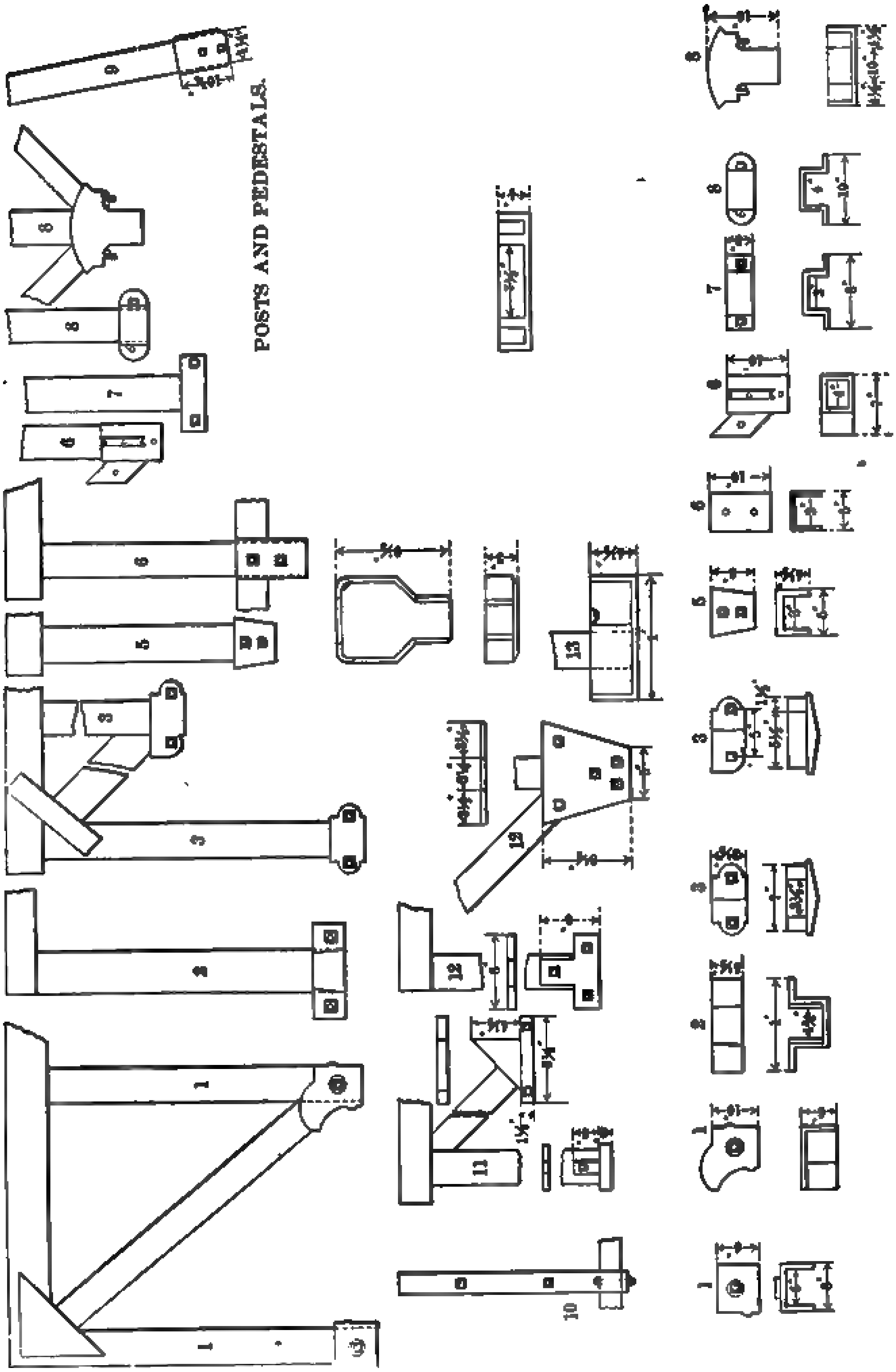


FIG. 99.

given requirements. They are of steel plate ¼ inch thick, with their sides and back end in one piece. The bottom plate is fastened to the sides by iron angle-arms. The standard coal-cars are of the shape, size, and details as shown in Figs. 96 to 100.

The metal cars are supported on one four-wheeled truck. The wooden cars, on two trucks, are almost all of them arranged with a swing-door at the end and provided with a trip-catch for a quick release. It is rare for companies to make their own cars, though all make their own repairs.

The Wheels are as large as circumstances will permit (the larger the wheels and the smaller the axles, the less is the friction). The wheels may revolve loosely on the round or the square axle, or they may be fixed to the axle and revolve with it. Some are capped with a recess in the hub, to receive the collar on the axle, and thus prevent admission of grit (Fig. 100). They may be "inside" (below) or "outside" (beyond) the body of the car. As to the relative merits of the inside and outside, or loose, wheels, it must be admitted that engineers are not unanimous, though the former have the larger number of adherents. Outside wheels are more easily oiled, are cheaper, and admit of the body of the car being set lower down; they do not run as smooth or last as long as those fixed under the body of the car. Loose-wheel cars may be better for short roads with sharp curves, but they are harder to pull. With fixed wheels, one of the mutually dependent wheels, in travelling about curves, must slide. For this reason, and because they are easily lubricated, loose wheels, or cone-fixed wheels, are preferred by many. A great many mines have abandoned loose wheels after careful trial. At the Drifton anthracite mine a compromise is effected by using a pair of fixed and a pair of loose wheels.

The coal-car wheels are of cast iron, between 16 and 18 inches diameter, and those of ore-cars about one half that and solid. The former have hub and arms to allow of "spragging."

The wheel-base of the car-trucks is a trifle greater in length than the width of the track. Its width is ¾ inch less than the track gauge.

The life of a car depends upon conditions too numerous to detail here, but it may be given as averaging three years in coal-mines and a trifle longer in metal-mines. They all grow larger with age. The wooden cars become loose and swell; the iron ones are quickly battered out of shape.

Brakes are provided on all coal-cars which go to the face of the workings. Otherwise they are not required.

Car Resistance.—The frictional resistances offered by the wheels at their circumference on the track and at their journals amount to about 2 per cent of their total weight. This quantity may be smaller where care has been given to their construction, as, for example, those which are employed in connection with the electric mine-haulage. This friction can be reduced by the liberal use of lubricants and care in the finishing of the journals. If the latter are boxed and kept free from dust, the frictional resistance will be materially reduced.

The lubrication of the bearings may be obtained in the ordinary way by the use of oil-soaked waste in the boxes forced against the journals. Self-oiling journals would further reduce the friction. They are inexpensive, and are desirable if saving in power is to be considered. By their use one third of a pint of oil will last from two to four months, according to its quality and the distance traversed. The saving in power is $10 per year per car, compared with the inefficient methods. This includes also the cost of axle replacements. Self-oilers are also economical of lubricant, as compared with the crude method ordinarily employed of pouring oil into a box and bearing. The oil used is usually one of the heavy varieties of lubricant with the addition of some axle-grease.

Lubricants.—An efficient lubricator must have sufficient body to keep the surfaces between which it is interposed from coming into contact. It must have the greatest fluidity consistent with its needed body. A small coefficient of friction is necessary, as well as a maximum capacity for conducting the heat. It should be free from any tendency to oxidize or gum, and free from acid.

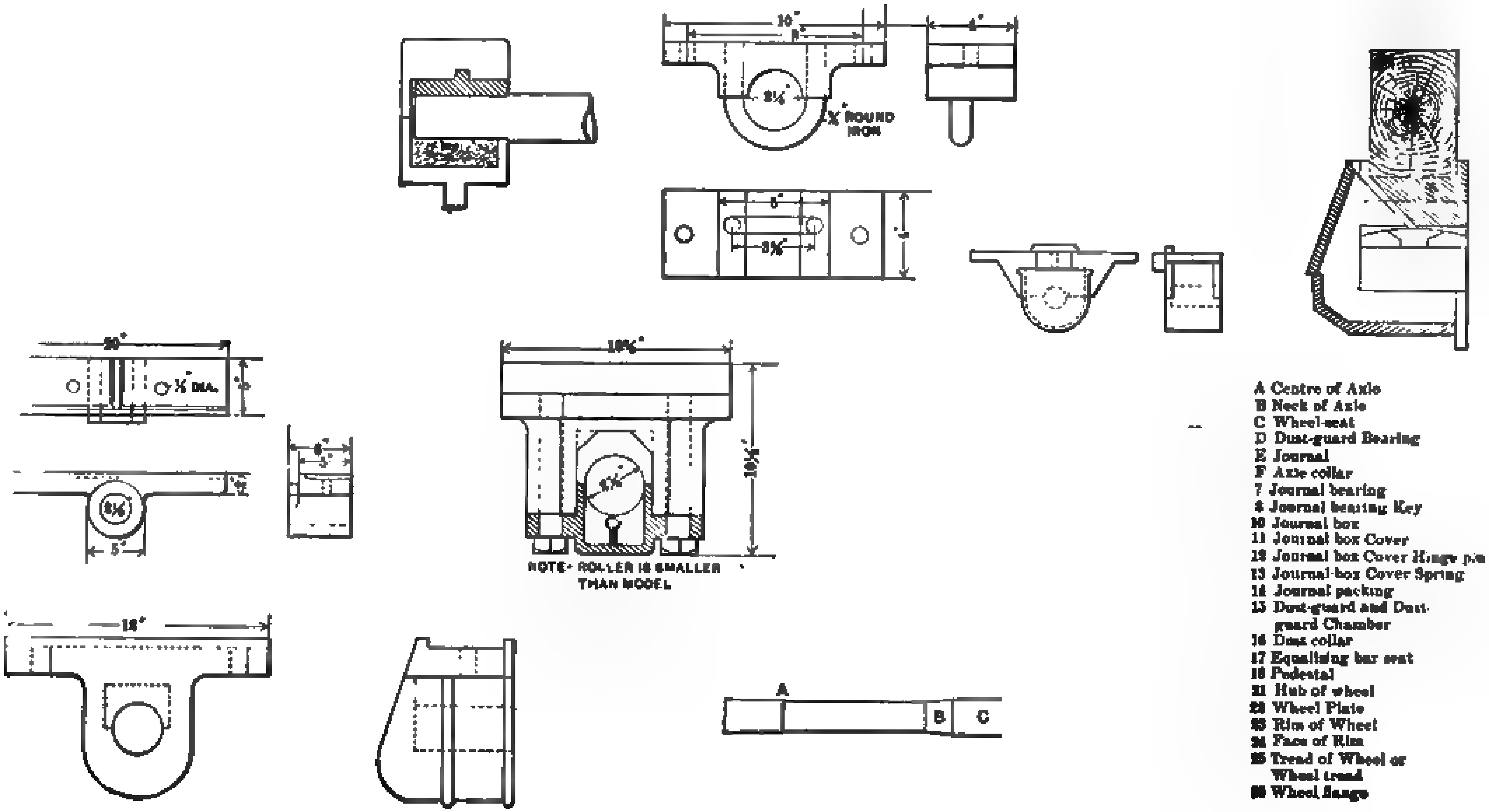


FIG. 100.—Details of Journal-boxes.

Testing Lubricating-oils.—The engineer should have means for testing, as oils intended for lubricating purposes are often adulterated with foreign and even injurious matter. A hydrometer and flash-test cup are sufficient for this purpose.

THE FLASH-POINT AND SPECIFIC GRAVITY, BEAUMÉ, OF LUBRICATING OILS.

	Specific Gravity.	Gravity, Beaumé.	Flash-point, Deg. F.		Specific Gravity.	Gravity, Beaumé.	Flash-point, Deg. F.
Castor-oil (pure).	0.9639	15	......	Tallow-oil........	0.9080	24.5	540
Linseed-oil.....	.9299	19	505	Heavy engine-oil..	.9032	25.5	414
Cottonseed-oil...	.9210	19	518	Castor machine-oil	.8919	27	324
Lard-oil........	.9175	23	505	Light engine-oil...	.8917	27	342
Rapeseed-oil....	.9155	23	......	Sperm-oil........	.8815	29	478
Cylinder-oil.....	.9090	24.5	575	Spindle-oil.......	.8588	33	312

The Hydrometer.—All oils are lighter than water and their purity may be ascertained by the use of a Beaumé hydrometer. If a contract calls for a heavy engine-oil of 25° Beaumé, the hydrometer at once reveals whether the same is received. If adulterated, a lower Beaumé reading reveals the blending. The adulterants used are, of course, cheaper than the pure oil. A "lard-oil," for example, of 23° Beaumé, sp. gr. 27, must have been adulterated with a lighter oil, and the only cheaper ones possible are the light petroleum or the neutral oils.

Viscosity Test.—The viscosity and gumming tendency of oils may be detected by noting the time required to traverse a given distance down an inclined plane, and the time when this ceases. The oil remaining limpid the longest has the lowest gumming tendency, and that which flows the farthest is the most viscous. A nine-days' test of oils gave the following results: A heavy mineral-oil, for engine use, traversed 87 inches on the ninth day; common sperm-oil, 63 inches on the ninth day; olive-oil, 22 inches on the ninth day; linseed-oil, 18 inches on the seventh day; and lard-oil, 12 inches on the fifth day. In each case the day given is that on which the oil ceased to travel.

The simplest method of testing the fluidity is to dip blotting-paper into the oil and hold it up to drain. An oil of good fluidity

presents symmetrical drops. One of high viscosity spreads readily over the paper. The rate of gumming may also be ascertained by its retention on the paper, while at high temperature, for some hours.

A Ring Test.—An oil which is composite in nature will show on a clean sheet of blotting-paper two or more rings, the outer one indicating the poorest oil. The central defined ring, dark in color, is due to the heavier oil. If the rings disappear, it is mineral-oil. Paraffine stock in the oil will be revealed by a well-discerned ridge near the centre. A light hydrocarbon oil does not give any permanent translucency to the paper.

The Flash-point Test.—This consists in finding the temperature which an oil will endure without being vaporized and ignited. An oil for cylinders should have a high flash-point; those not intended for lubricating steam surfaces may have a much lower flash-point. The flash-point can be determined by heating a sample of the oil in a cup over an alcohol-lamp, on a tripod. A thermometer inserted in the oil will note the degree at which ignition takes place. Care should be taken that the thermometer touch neither the bottom nor the sides of the cup. As the oil reaches 300° F. a match or lighted twine should be passed across the surface of the oil until a blue flame shows the time of ignition. Lard-oil has a high flash-test. If the sample submitted burns below 500°, it is adulterated with cottonseed or petroleum; linseed-oil will have its flash-test reduced by the same adulterants.

Fire-proof Oils.—These are valuable on bearings inclined to heat. They derive their virtue from the fact that they contain considerable water, held in suspension in the oil by means of a mineral soap. When applied to heated bearings the water serves to absorb the frictional heat, leaving the oil for lubrication.

Light-running machinery is lubricated with oil of 300° flash-test, heavy machinery with oil of 400°, and engines with oil of 550°.

Oil intended for the bearings of motors and generators exposed to low temperature should be limpid and free-flowing.

Cylinder lubrication is best obtained from a petroleum-oil of

high flash-test (530°) and viscosity. It is compounded with 5 per cent of tallow-oil if the steam is wet.

The air-cylinders of air-compressors and of gas-engines should be lubricated only by a pure petroleum-oil.

Geared wheels are oiled by some preparation composed of tar, wax, and similar substances combined with rosin, which, when applied to the teeth in a molten state, will cool to a tough, elastic coating, reducing friction and noise.

Handling Cars at the Tipple. — To facilitate the delivery of the mineral at a minimum expense, and with as little handling as possible, the landing-station or the tipple is fitted with quick-dumping appliances for the cars and suitable grades for their prompt return. The level of its floor is below that of the mine mouth or the landing-station in the derrick, and far enough above the railroad track or the point of shipment to permit the mineral to have a continual descent between the time of dumping and the time of shipment. The tracks to the tipple are laid favorably for the load, and the character of the conducting line between them will depend upon the distance and the slope of the ground. In the case of a slope mine the loaded cars or the train can be hauled directly to the tipple and returned thence by gravity. Any grade exceeding one per cent will allow the empty car to return.

Dumping Cars.—The dumping of cars may be accomplished by opening one side or the end of the car by inclining the car on its axle or by tipping it upon some form of cradle. Mineral which is to be subsequently broken or sorted is dumped upon a screen or the grating to the sorting-floor. Some tipples are provided also with self-indicating weighing-machines, by which, as the car is emptied, the machine registers its weight, turns over, shoots out the coal, and returns to its normal position for another load.

The iron-body car of metal-mines is set on a swivel near the centre and held in position by a lever. When it is to be emptied the lever is unhooked, the car turned to position, the door raised, and the car tilts because of the slight overhang at the discharging

end. It empties without further effort from the trammer (Fig. 101). It is fastened by the latch as soon as the car body has been

Fig. 101.—A Steel Ore-car.

swung into position on its truck. Cars intended for tunnel use empty at the end, the door being hung by strap hinges from

Fig. 102.—A Side Dump-car

an iron rod at the top. It latches in the same way as mentioned. Another variety of tunnel car consists of double iron frame,

pivoted at the middle of the top line on the side. It is on two trucks for dumping. The latch, located at each side, is raised, the car opens at the middle and empties between the trucks (Fig. 103). The doors on the ends of coal-mine cars are locked by latches similar to those of the iron-frame cars. In Figs.

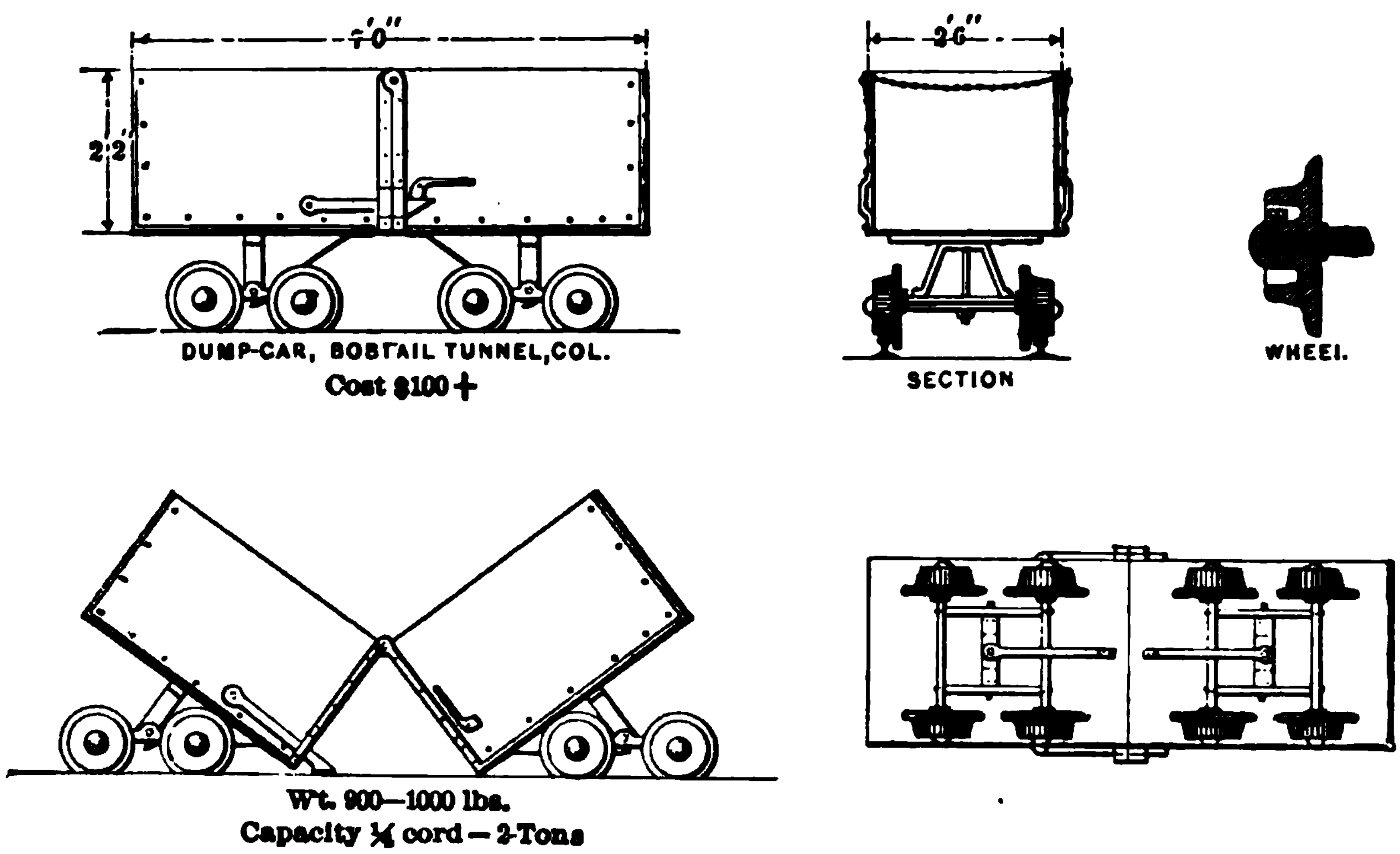


FIG. 103.—Tunnel Dump-car.

105 and 106 are illustrated some typical latches used in the coal regions.

The empty cars can be returned to the mine by being raised by a short chain-haulage to an elevation sufficient for them to return by gravity to the mouth of the mine. A chain-haul can be used to return the empty cars to a "kick-back" beyond the shaft if the distance from the dump will permit. This is then worked like the Ramsay caging apparatus, but is much cheaper than the latter.

Cross-overs.—Various combinations of tracks and machinery are employed to expedite the loading and unloading of cars on the cage and economize time at landing-stations. The plan is to displace the empty car upon the cage by one to be placed thereon. For example, the loaded car pushed by hand upon a cage bumps

off the empty car at the bottom of the shaft, or the empty car to be lowered into the mine is made to replace the loaded car, which is simultaneously run off. The automatic cross-overs are of several types and very effective. The usual plan consists in having the car approaching the dump release dogs holding the empty car, and allow it to pass over the dump to the "kick-back" (Fig. 104), whence it may be returned by the track on the left.

The Ramsay Caging Apparatus.—This consists essentially of two steam-rams placed back of the shaft and two transfer-tracks, running on a track across the tipple in the rear of the head-frame, which are operated by a steam-cylinder. Its operation is as follows:

FIG. 104.—The Ramsay Caging Apparatus.

After a car has been dumped it runs back by gravity past one side of the head-frame into one of the transfer-tracks, which is then moved by means of the steam-cylinder to the rear of the compartment where the next loaded car is coming up. When the cage with the loaded car is at the landing, the empty car is pushed by means of one of the steam-rams against the loaded car, which is then taken off the cage and the empty car left on the cage ready to descend, so that the loaded car is shoved from the cage and the empty one pushed on at one operation. When one transfer-track is back of the frame with its empty car, the other track is in position to receive the next empty car as it comes from the dump.

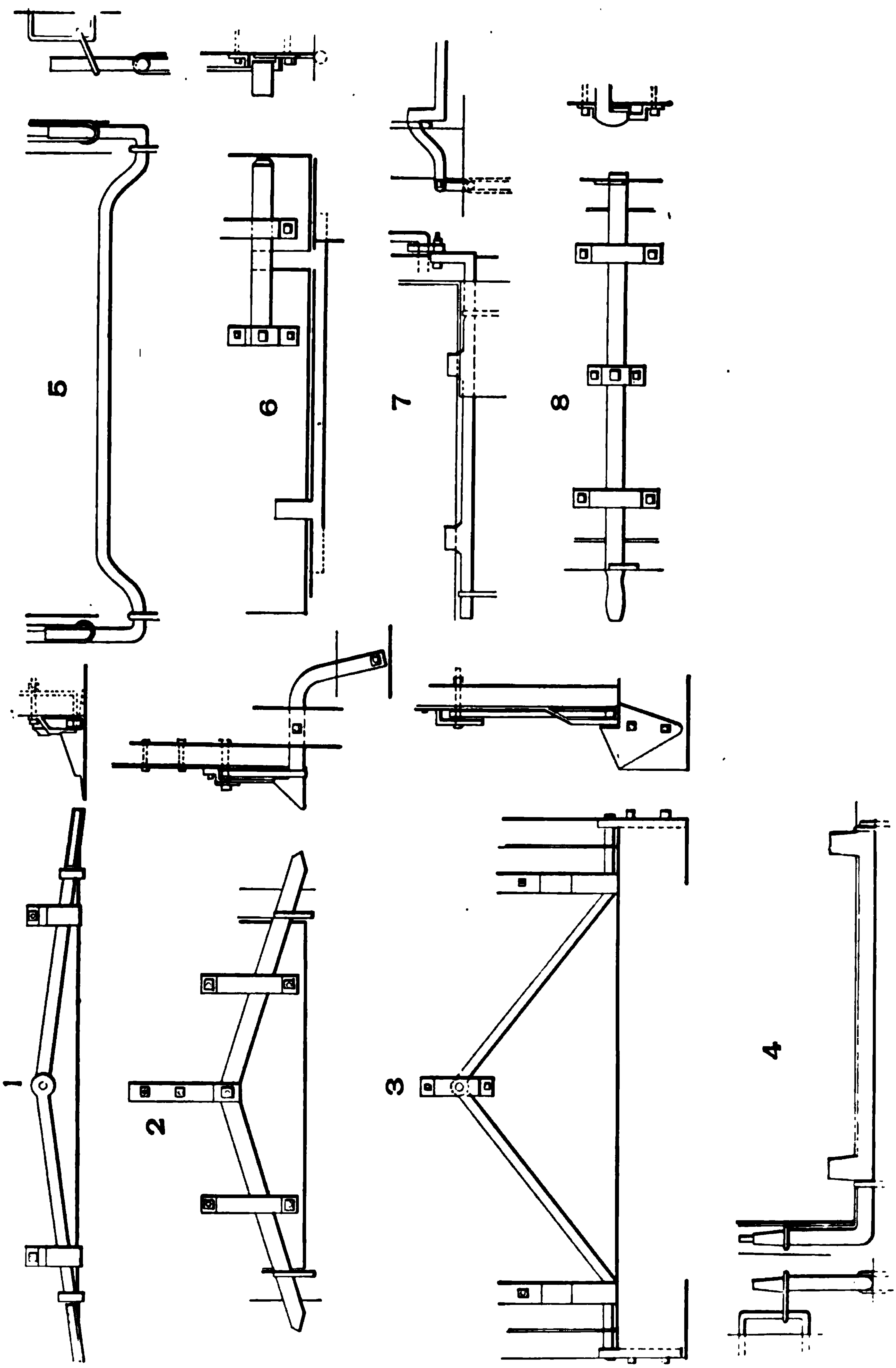


FIG. 105.—Details of Latches.

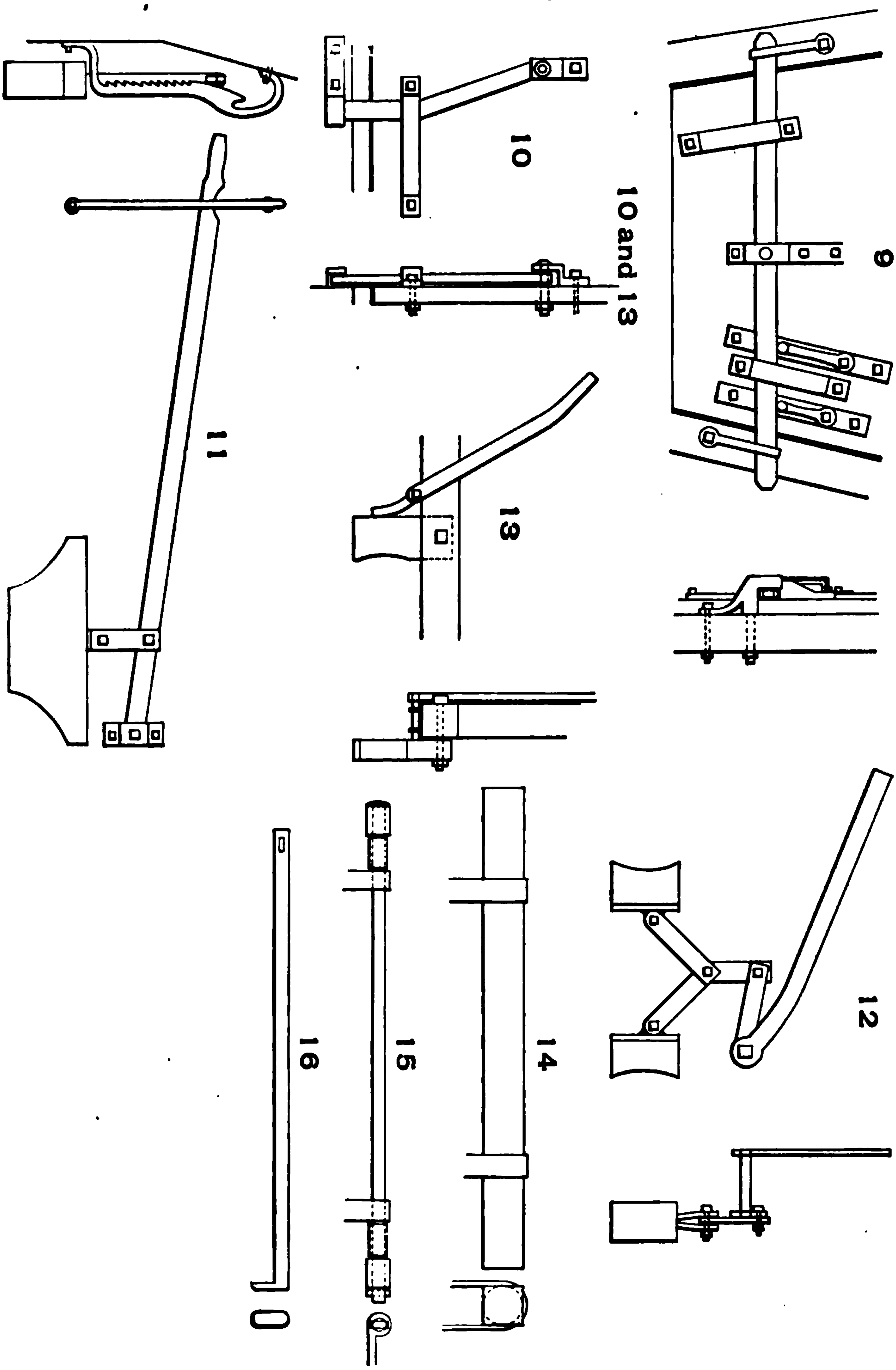


FIG. 106.—Latches for Mine-cars.

Cradle-dumps. — Various forms of cradles are provided at dumping-stations to facilitate the emptying of cars with or without hinged doors. These consist of some form of balance-frame upon which the loaded car is run and held. When the catch is released the frame, with its car, tilts and empties the contents as soon as the centre of gravity has passed slightly beyond the pivot, or axle, of support. Another form (Fig. 107), which is the

FIG. 107.

Behr device, is much used. The car is held on a platform and revolves with it around the point of support, as indicated by the dotted lines. The car in this device is almost completely inverted and thus emptied. Another form of cradle, used with

larger cars having no swinging doors, consists of a skeleton barrel having tracks in such a position as to receive the car, which is then completely enclosed. When the trap is released, the barrel is revolved by gear-wheels, and the car is inverted. In the anthracite mines are employed different methods of dump, as illustrated in Figs. 108 and 109. The shock of the loaded car running upon the truck carriage supported on the gravity tracks causes the front wheels of the latter to roll down the curve, while the end wheels are elevated and the car is tipped sufficiently to empty its contents, the door having been previously released.

Means of Access for the Men.—For purposes of ingress and egress mines are provided with ladders or man-engines, where the cage or bucket is not used. The statutes of the several States contain various requirements for the accommodation of the men. Some require the maintenance of substantial ladders in a separate compartment as the sole means to be used by the men for entry and exit. In other States operators are relieved of the necessity of keeping up a ladder-way, if safety-carriages are employed. The laws of many States forbid the use of buckets by the miners, while the general tendency in all regions is to insist upon two well-equipped escapement-ways.

If the angle of entry is below 30°, no special provision is necessary. The mudsills of the timbering break the descent into sufficiently convenient steps. For steeper angles, up to about 60°, some variety of tread is necessary. When the pitch exceeds this the compartment must be provided with ladders isolated from the hoistway. They should be inclined, uniform in direction, at an angle of not less than 10° from the vertical, to diminish the fatigue of climbing and enable the men to carry tools with them. At equal distances down the ladder-way (20 to 40 feet down a vertical shaft, and at greater distances on an incline) platforms are built of 2×6 beams and 2-inch planks, closing it, except for a manhole, at the foot-wall end. The ladders extend up through the manhole, and are fastened by staples or toe-nailed to the shaft-timbers, and rest on the far side of the plats. They are made of 2×6 standards,

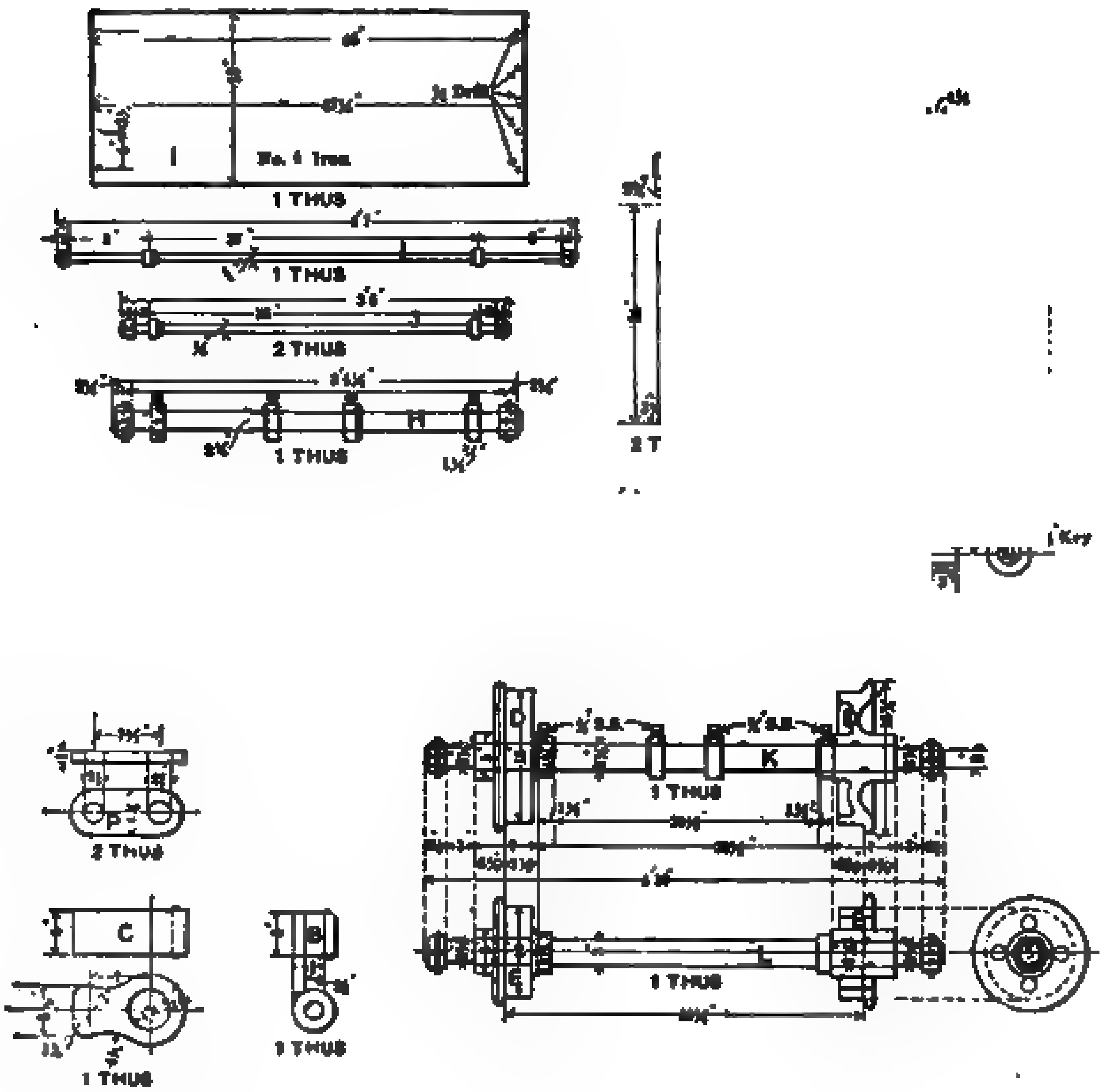


FIG. 108.—Anthracite Car-dump

18 inches apart, with iron or wooden rounds or rectangular slats 12 inches apart. The last-named are cheaper, last longer, and

SIDE ELEVATION

END ELEVATION

FIG. 109.—Anthracite Car-dump.

give better toe-hold than wooden rounds, which, in turn, are easier to use than the more durable iron.

In slopes having an inclination of less than 60° steps and rails are provided, the tread being 6 inches and the slope dis-

tance 12 inches. The headroom provided is at least 6 feet vertically; when the pitch is less than 45° the rule obtains making the rise plus the tread in inches equal 24; below 20° pitch nothing is required except occasionally a plank.

Though used in Europe for 1200 to 1500 feet depth, and in this country in deep mines, ladders are certainly not advisable. According to the Cornwall Society, their use deranges the respiration and shortens life by ten years. The miners reach the workings more or less exhausted, and the operators have lost the benefit of a proportionate amount of energy. Unquestionably, an element of success worthy of attention by mine managers—a pecuniary as much as a humanitarian question—is the proper treatment of and the conveniences for the men, who unconsciously reciprocate in an equivalent of work. Ladders waste time. It takes 15 minutes to go down 300 feet, and the ascent is twice as slow. A shift of forty men, following one another at intervals of 8 feet, entails a loss to the company of 31 minutes each shift. With buckets and cages the loss is not so great; eight men at a time, lowered 1200 feet, consume 40 minutes for every shift of 100 men. An additional loss occurs at tally-time from the reduction of the hoisting capacity, which, with the impatience of the men, leads to the crowding of the cage; but in most States the limiting number of men permitted on the cage is named.

The Man-engine.—Movable ladders or man-engines, invented by D'Orrell, of Clausthal, were adopted as acceptable substitutes for ladders and were once popular in deep mines. When introduced into Cornwall by Mr. Lorn, the engine was declared by the Royal Polytechnic Society a very "great boon to miners." Its introduction involved the addition of some machinery, and, though easy to operate, it is now obsolete as an economic arrangement.

Two rods, of decreasing cross-section from the top down, receive at the surface an oscillatory motion from balanced bobs, operated by an engine having a fly-wheel and other regulators. The dimensions of each rod at any point must be such that it will have

the requisite tensile strength to support the weight of the part below it, loaded with men. The rods play between roller-guides 50 feet apart, and are provided with wings and catches, after the manner of the Cornish pump-rods, which may, in fact, be utilized as "Fahrkunst" rods without much extra power.

Each rod has a small platform (Fig. 110) about 12″×12″ or 18″ at every 12 feet—double the length of the stroke. A handle 4 feet above the platform gives support to the miner, who is carried up 6 feet on one rod, which brings him opposite a platform on the companion rod; upon this he steps, to be lifted 6 feet more, to meet a plat on the first rod, which has been coming down to receive him. A miner stepping from one to the other is carried up or down at a rate of from 48 to 96 feet per minute (each rod makes 4 to 8 double strokes, delivering one man each time; those at the Calumet and Hecla make five strokes). As there is no limit to the depth to which these may be carried, and as they are capable of working alike in slopes as in shafts, it is not surprising that they "take" so well. They replace hoisters, and require little additional power or space. Tools and supplies cannot be carried by the miner, but may be delivered by the cage or bucket.

A single rod is also used, its companion being replaced by stationary platforms attached, 6 feet apart, to the shaft-timbers. Upon these the ascending men wait during the down-stroke of the rod. The single-acting man-engine requires chains and counterpoises at intervals, to balance it and to prevent the shock incurred at the end of the stroke.

FIG. 110.

From the fact that a misstep would be fatal, it would seem as though man-engines were extra hazardous, yet the accident record does not confirm this belief. Some confusion is caused

by a man missing his plat and riding on, to the annoyance of those following him; but this is of rare occurrence unless his light goes out, for there is a halt of several seconds at each change of motion. In Prussia, out of an average of 100,000 men employed for ten years, only 57 were injured on the man-engines; in Cornwall, 17. This is more than compensated for by the increased length of life of the miners using them.

The cost of machinery, etc., for a 1200-foot man-engine is $18,000, upon which interest and depreciation may be figured at $2500,—amounting to 10 cents per man daily on a gang of 100 men. The running expenses at the Dolcoath mine are 4 cents per man for 2400 feet.

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CHAPTER VIII.

UNDERGROUND HAULAGE SYSTEMS.

The Extent of Haulage Requirements.—In any coal-mine of moderate size there are fully 500 miles of track and headings into the rooms, representing perhaps 2000 miles of road, which at some time during its history had been laid down and given service. It has been estimated that in the production of 200,000,000 tons of coal at least 5000 miles of track extension are made each year. The great amount of it indicates the necessity for giving special attention to its construction. The cost of delivering coal from the working face to the tipple when the tracks are level and the distance not over a mile is between 6 and 10 cents per ton. During a single month an average of thirty-one mines using rope haulage showed a cost of 7.5 cents per ton. In twenty mines employing electric haulage the run-of-mine coal cost 6 cents per ton, and 8 cents in an average of twenty-one other mines having mule haulage. Probably, over the United States, the average cost of animal haul is nearly 15 cents. The cost of haulage increases materially as the distance of travel exceeds 3000 feet. Beyond this length of haul it may be advisable to sink another shaft if a convenient point of shipment can be obtained at the surface and the depth of shaft be not too great.

The Grade and Direction of a Haulage Road.—A haulage road is as straight and direct as circumstances will allow, so designed as to utilize the natural forces to the closest degree. The shaft and slope are carried down to the lowest point, that to it may be delivered the mineral from the working face on a

continual descent. Dropped through the mill-holes from stope to cars, the ore is moved by hand- or horse-power to the foot of the shaft; and coal, leaving the face in cars, should have the advantage of the down-grades in its delivery to the shaft bottom.

From the workings to the haulage-way the inclination of the track will depend solely upon the mean inclination of the floor, but the cars should be capable of moving by gravity without the use of blocks or sprags. A sprag is a billet of wood inserted between the spokes to prevent the wheels from revolving, thus converting rolling friction into sliding friction and checking the speed of the car. On a 3 per cent grade only one sprag is used, but on a 10 per cent grade four sprags will be necessary to keep the speed of the car within moderate control. A grade exceeding one in five is positively dangerous. Roads are often designated as one-, two-, three-, or four-sprag roads, according to the number of wheels which must be blocked.

The best grade for any haulage road is the gradient of equal resistance—that on which the work expended in hauling a loaded car down equals that of the returning car up. A steeper grade than this is wasteful of power both ways. As evidencing the influence of grade in the reduction of useful power, it may be cited that a locomotive can pull up an incline of 2 feet in a hundred only 80 per cent of what it can haul on a level, and on a 4 per cent grade but 30 per cent. This is correspondingly true of animal haulage. While a mule can haul three loaded cars on a reasonably level track, it can haul only one on a 6 per cent grade.

Haulage-ways in Metal-mines.—Metalliferous mines rarely present any difficulty for the road engineer. Very few roadways feed to any given landing. In a vein there is but one line on either side of the shaft for which provision is to be made. In large ore bodies only a few tracks will radiate from the point of outlet, and but few cars will travel over each.

Any desired grade may be laid in vein mines. It is as much greater than that required for drainage as the available motor power will allow for returning with the empty cars, or controlling

the speed of the loaded cars out. The direction of the roadway is fixed by the sinuosities of the vein wall. For animals due regard must be had to economy of size and weight of car and its contents; hence the grade will probably not exceed 1.5 in 100 feet. For locomotive haulage the grade may be carried to 4 per cent as a maximum.

Haulage-ways in Coal-mines.—Here the engineer must accommodate himself to the stratigraphical conditions. The thickness of the coal-bed and the frequency and amount of the existing faults, folds, and inclinations must be reckoned with, and a uniform grading would demand their elimination, according to the character of the enclosing rock.

In the main headings the hillocks may be removed and the sumps in the floor filled, or the roof material over the depressions may be ripped down and the sumps filled with the waste. The latter is more economical. It is far more expensive to take up a foot of bottom than to blast down 3 feet of thickness in the roof. Again, the cost of maintenance is not so great where the road-bed filling is of loose slate from the roof, as in the customary floor of fire-clay or limestone. Often the soft floor of the coal-seams cannot be disturbed, and it may not be desirable either to fill up the depressions or to remove the hills, because of inconvenience in making connections to rooms. It is probable, too, that the cleavage planes in the coal will determine the directions of the haulage-ways, and the average dip of the vein will determine their grade.

In coal-mines there are always two parallel headings for ventilation purposes, the engineer having his choice as to which shall drain the water and which be used for haulage. It is rare that the same heading would be required to give both services. Usually the intake is the one employed for haulage, because of the better quality of air and, in consequence, a less cost of maintaining the track. This roadway is never obstructed by the air-crossings, which usually are placed on the return-airway. When no other condition determines the choice, the roadway higher in elevation is employed for haulage and

the lower one for drainage, with frequent communications between them to allow of the accumulated waters of the former to be promptly carried to the latter.

The haulage gangway is of single-track width only when the system employed is the tail-rope or the locomotive. The double-track gangway is employed where the traffic is heavy, or for an endless-rope haulage-way. The advantage in the latter is the great area given to the air-current, and also the opportunity afforded for utilizing the excess of the power in the down-going car or train to assist the engine in raising the load on the other track.

The requirements for ventilation demand a roadway of liberal area which will permit a large quantity of air to pass through with the minimum of friction, and also reduce the effect of the disturbance to the current by the passage of the train through it.

The enlargement of the area necessary to provide for the cars may be obtained by increasing the width or the height of the roadway. The former is preferable unless the roof is very firm and the bed quite thick. The objection to it, however, is the increased length of sills and the timbers necessary to maintain a good level on the ordinary floor of the coal-mine. It is preferable to increase the height unless the roof will not bear cutting down. It is a cheaper method, for the dimensions of the sills would be only of moderate size, and those of the posts would not be any greater than for a wide roadway.

Track Construction.—The tracks on the haulage-ways are constructed with a view not only to diminish the working expenses, but also to maintain a large capacity with a given amount of power. The difference in cost and the time of laying between a heavy rail on a substantial track and one which is poorly laid is not large, nor is it commensurate with the difference in their efficiencies. A reduction in the grade of ½ per cent saves annually in track maintenance an equivalent of 30 tons of fuel for motor power for each car running through the mine. The cost of such construction might, however, prove prohibitive of any operations. The most that can be done is to improve the roads

within reasonable limits of cost after careful surveys and levellings of all roads intended for main arteries.

The road-bed is made as elastic as possible by the use of sand, dump-coal, or ashes, without being too soft for support. The choice of ballast depends upon the character of the mine floor. If the latter is inclined to be wet or rough, then rocky material would be preferable to sand or ashes. When the floor is very strong, clay can be used to advantage with fragments of rock grounded in. The dry road-bed can be secured by rounding off the surface between the tracks to a slight arch. This will assist the removal of water from the track to the drains.

The ties are laid 2 feet apart between centres for rails of 15 to 30 feet in length. To these the rails are spiked, four spikes being used on each tie. Care is taken that they are not all driven near the centre of the tie, causing it to split. For all rails of 16 to 20 lbs. per yard the ties are 5 inches on the face, 4 inches deep, and 5½ feet long. For the heavier rails in the mine they are 6″×5″ deep. They are laid normally to track, and on curves are laid in the direction of the radii of the curve. In some mines the ties are creosoted to increase their durability.

The rails are the common steel T rails, having the same specifications as are adopted for railroad service. Their weight varies from 12 lbs. per yard in the rooms to 40 lbs. per yard in the slopes. They cannot be too heavy within reasonable limits. The greater the pitch of the slope and the greater the speed of haulage, the larger and heavier should be the construction of the track. The rail for locomotive haulage-ways requires a weight of 10 lbs. per yard for each ton that falls upon the drivers. The cost of one mile of 16-lb. rails is about $1600.

The track in the room and at the long wall-face is laid as rapidly as the mining operations progress, and with as great care to secure a firm run as in the main ways. It is often laid in sections composed of two rails of 15 feet each. These sections are portable and carried up to the face of the workings with the progress of the room, the expense of such track-laying usually being included in the price paid to the miner per ton of coal.

Their advantage lies in their simplicity, strength, and durability. The gauge does not depend on the skill of the workman, and will be preserved as long as the line lasts. The lightest rail advisable for such purposes is that which weighs 16 lbs. to the yard. Wooden rails are no longer employed even for temporary service in the rooms.

The Gauge of Track is rarely less than 30 inches or more than 44 inches. Local conditions determine the choice. A broad gauge gives a greater stability to the cars and a reduction of haulage expenses, but is more costly to build. The minimum gauge of 30 inches affords easy haulage, opportunity for sharp curves, and a cheaper track.

Turnouts.—The sidings are formed by a set of points and crossings, either right- or left-hand, a section of curved line, and a number of sections of straight line.

Switches from one track to another are obtained by the use of movable points similar to those employed on the surface roads, though shorter in length. The point is simply a spike driven into one of the ties supporting the rails. It is held in position by and turned upon an iron pin passing through the plate welded to the base of this point and thrown out of line to the branch side when the car is to be switched. When two of these points are connected by a flat piece of iron passing under the rails of the track, the pair can readily be thrown right or left by a bell-crank lever, and both rails of the track are opened for the switch. Wherever possible single sets of points are arranged for, the wear being less on two pairs of single points than on one set of double points, besides which single sets are simpler to lay down and keep in order. The use of the three-way points and crossings for two opposite side branches are avoided underground by turning the right- and left-hand lines from the straight track in positions which are not quite opposite to each other. The points and crossings are made of standard sizes for curves of any special radius required to suit the gauge of the rails and the wheel-base of the engines. All frogs and switches at the shaft-landings or slope bottom are of pointed sections of standard rail sizes,

manipulated by a spring or a hand-lever. For locomotives on a main tramway the curves should not, as a rule, have a radius less than from 100 to 125 feet.

When one rail intersects another, frog crossings are used. They are either plain rail intersections, broken at the points of crossing, or they are the standard frogs furnished by manufacturers. The points, rods, and levers used underground are usually made and fitted by the colliery blacksmith, the plate-layer assisting in the fitting.

In passing from a heading or entry to the room a "parting" is used. This is similar to a switch, the wings running backward from the frog end near the rails, but are cut in a slanting direction in order to provide a sufficient clearance between the point and the rail to allow the wheels of the car to pass along the main track when it is not to be deflected into the room. There are two forms of partings, one of rails and another a combination of rails and cast-iron pieces, the latter being much inferior to the former. The former has less pieces, gives more solid track, and is less liable to derail the cars. It is cheaper and can be made in the blacksmith-shop.

Turnouts are obtained by the use of plates or turntables. Of the former the simple iron plates laid on stout planks revolving around a vertical pivot are employed where animal power is used. These are common in metalliferous mines, being preferred to self-acting switches. Not infrequently is seen at intersections of roads a large fixed iron plate, raised upon suitable supports, level with the road-bed, and having curved guide-rails at each of the four corners. These may be employed as turntables by dragging the car around on the plate until it faces the desired track, when it is run off. This crude device serves well, but has nothing to commend it.

Curves.—Curves in the track are built in the same way as on the surface, the radius of curvature, however, being very much shorter. Connecting two entries on the main haulage-road, the minimum radius for narrow-gauge tracks is 60 feet, and for a gauge of 45 inches 100 feet. At the shaft-landings the

radius is as short as 30 feet for 36-inch gauge, and 50 feet for 44-inch gauge.

The Degree of Curvature of a Curve.—By this term is understood the number of degrees of circular arc measured at the centre which will be subtended by a chord of 100 feet length. Thus a 10° curve is one in which a 100-foot chord at the circumference is comprised between two radii making 25° at the center. The radius of a one-degree curve is practically 5730 feet. Hence the 10° curve will have a radius of 573.0 feet. This is only approximately correct, but is sufficiently accurate as a standard of measure for engineers.

All curves on underground roads are laid by transit with care equal to that employed on the surface for larger work. The central line between the rails of each track is staked out at intervals of 20 feet, from which on the other side are laid down the two rails. The latter are now bent to conform to the curvature of the circle on which the rails are laid.

Elevation of the Outer Rail.—Provision is made for neutralizing the effect of centrifugal force, or the pull of the rope, on the curve by elevating one rail an amount depending upon its sharpness. On curves used by locomotives the outer rail is the one which is elevated, as also on the slope haulage-ways when the weight of the cars draws the main rope off its drum. In a slope haulage the outer rail is raised 1 inch on a 9° curve, 2 inches on an 18° curve, and 4 inches on a 36° curve, the latter being the maximum. It never exceeds 4½ inches in height on a curve of 50 feet radius, or of 114° curvature, when the gauge is 42 inches. This would be suited to the ordinary speed of locomotive haulage, or that of cars running down the slope by gravity.

The inner rail is elevated above the outer one only on such curves and under such conditions as those in which the action of the centrifugal force is less than that of the pull of the deflected rope. In this case the rope passes over rollers near the inner rail and tends to pull the car inward. It is possible, by comparing the value for centrifugal force under given conditions, as calculated below, with that of the pull of the rope upon the car, to

determine the amount of the inclination which must be given to either the inner or the outer rail.

Let g = gauge of the track in inches;
e = elevation of the outer rail in inches;
R = radius of track in feet;
v = velocity in feet per second;
D = degree of curvature of centre of track;
c = centrifugal force due to the speed around the curve.

Then $$e=\frac{0.031gv^2}{R}=0.0000054v^2Dg.$$

Conveniences at Landings.—Whatever may be the system of haulage, arrangements are made for the prompt handling of cars at the shaft bottom. When the coal is hauled from the dip by rope or locomotive, the slope is continued, if possible, on the same side of the hoisting-shaft at a convenient distance from it, where the requisite number of loaded cars can be accumulated and allowed to run by gravity on a one per cent grade to the shaft.

The empty car can be run off the cage on to a short down-grade of about 4 per cent, and, with the momentum given it by the loaded car being run on the cage, would descend far enough to rise up on an opposing grade of about 2 per cent to an automatic switch, which delivers it to the empty-car siding. Here it may be taken up by the locomotive, or a rope passing around the shaft through an entry designed for this purpose. Sometimes a short chain haulage operated by an electric motor is employed to draw the empty cars to the main haulage-way when the gravity system cannot be employed.

The landings for the cars in the mine at the end of the endless-rope haulage-way are reached by switches from the main track by slide-knuckles, a place being allowed for the rope to pass through the switch without injury from the flanges of the wheels passing over it.

Safety Devices Along Haulage-ways.—To prevent accident from runaway cars on a grade, whether underground or at the surface, automatic devices are employed either to check the speed

of the car or to automatically shunt it off the track. Cars are frequently fitted with a drag projecting below the car, which, when the latter breaks away from the hoist-rope, catches in the floor of the track and prevents the runaway. A similar device is employed whereby a car trips a block which is thrown across the track before the car reaches it and stops the car, preventing further damage being done by its descent to the lower end of the plane.

Catches at the Top of the Incline.—These are to prevent cars from breaking away at the top of the plane. Reaumaux's catch consists of a bent lever with two unequal arms turning around a vertical axis. It is placed flat on the ground and is parallel to the tramway and at such a distance that at least one of the ends always bars it, with the short arm next the side of the inclined plane. As the empty car reaches the top of the incline and passes forward its front wheel touches the long arm after the hind wheel has passed the short arm, and the way is blocked. Another run on the same track cannot be made until the protruding arm is turned back by the planeman.

Blocks or stops are simpler appliances. There are two pieces of wood at right angles to one another moving on an upright pin. One arm is thrown across the track to hold the car in place until the latter is to be released and lowered (6, Fig. 84). Then it is knocked out. The same plan is adopted in 10, Fig. 84, by the weighted lever.

Stops Along the Incline.—To guard against accidents from runaways along the inclines a heavy iron bar is swung as shown in 9, Fig. 84. It interposes no obstruction on the track until some one pulls the wire and releases it, as shown by the dotted lines. A similar balanced block, Fig. 84, is used, the car being caught by the end, *b*.

The Mortier safety-catch consists of a movable axle with levers placed in the axis of the roadway and supported upon sleepers, 8, Fig. 84. In one extreme position it is opened by the axle of the rising car; in the other, it closes after the train passes. In the middle position both levers bar the passage The planeman can also move the catch by a pedal.

Catches at the Bottom of the Incline.—At the bottom of the incline are additional safeguards. A "seizer," shown in 12, Fig. 60, is placed at the foot of the incline. An iron bent lever-arm supported on a horizontal axis at the apex projects vertically above the rails between which the frame stands. A strip of iron swinging horizontally keeps up one arm, which is inserted into the link of the chain. Even if the top man should push his train over the plane before the bottom train is ready, the chain would not move until the upheld arm is released and allowed to fall into a recess, and thus free the cars. Various other provisions of a similar nature guard against the ill effects of a broken endless haulage-chain.

The Choice of Haulage Systems.—The choice of a system depends upon the grades of the haulage-ways. When the latter are horizontal, power will be required in both directions, and obtained from the horse, locomotive, stationary engine, or rope haul. When the grade is toward the shaft, the loaded train may be pulled with any of these powers The horse or mule is, however, used to advantage only on a favorable track of less than 2.5 per cent. The locomotive may be employed for grades not exceeding 4 per cent. Above that it is impossible for it either to pull a long train up the inclination or to check its descent. The endless-rope system or the tail-rope method is used when the grade does not exceed 20 per cent. Beyond this inclination the plant will be self-acting, for the loaded train going toward the shaft pulls the empty cars back into the mine. When the grade of the haulage-way is slightly against the loaded car, the horse may do for a short haul. If it exceeds 2.5 per cent, the locomotive can no longer be used advantageously. On a grade exceeding 5 per cent a traction locomotive is totally incapable of pulling any load. Up to a limit of 10 per cent, a track locomotive may be employed. For all steep inclines some rope system must be employed.

Tramming.—Man-power is employed for incidental work on the short hauls on level ground, or in rooms. The weight of car and degree of inclination for such power are necessarily very

small. The average man is capable of exerting a push of 27 lbs. at the rate of 2 feet per second. This would move a 2-ton car on a level at a speed capable of making from three to six round trips of a ton per mile. The condition of the roads and running gear will definitely fix this.

Animal Haulage.—Horses and mules are employed underground for limited purposes of transportation singly or tandem, according to the length of the trip and the weight of the train. It is customary to work them in traces; though in some mines shafts are used which give considerable holding-back power not to be had from traces. There can be no objection to traces on level or nearly level roadways. But with loaded cars downhill sprags must be placed in the wheels so as to act as a drag for trace-mules. The inclination of the roadway for animal haulage power is not over 1.5 per cent, as the mules rarely take the cars to the surface unless the inclination of the entries especially permits it.

The utility of the animal is confined to haulage in the secondary ways, to the rooms, and to switching where economy in height must be practised. Here the mule has the superiority over the horse because of its shorter stature with equal strength. A thickness of coal-seam of 3.5 feet will accommodate the mule, while one of 4.8 feet is essential for the horse. The kind and size of horse purchased for colliery working will of course depend upon the specific work for which it is required and the nature of the roadways, size of cars, etc. Before being introduced into the pit they are worked on the surface for three or four weeks to reduce the risk of importing infectious diseases into the underground stud. The possibility of new horses being up to the work expected of them may thus be ascertained.

The ordinary speed of the animal is two miles per hour. It has a tractive force of about 150 lbs., at which a single horse or mule may be estimated as an equivalent to from 8 to 9 gross tons of load per hour, averaging four round trips of a mile each per day. The condition of the track, the ventilation, and the amount of delay at the termini of the trip would reduce this

average and make the daily capacity of the horse or mule about 40 tons. This represents a gross yearly capacity of 9000 ton-miles. Under ordinary conditions one mule will serve for the haulage of the output of ten miners. A mine of 1000 tons daily output will require 90 mules, which with equipment will cost $8000.

The average cost of mules, as well as their keep, is nearly the same as for horses. Their care averages 60 cents per day, inclusive of feed. The daily allowance of food varies with the work done. It is not far from 18 lbs. of grain and 12 lbs. of hay per animal. A less economical fodder now used is "ensilage." This is fodder cut in the field in the usual way, but, instead of being dried, is chopped up small and stored green in brick or cement-lined pits, or silos. Dry food is a more convenient feed, but dearer than ensilage.

The number of horses under one ostler or keeper should not exceed ten. They are generally stabled underground, and many, once below, never see daylight again. Where the workings are at a considerable distance from the shaft, the stables are built well back in the interior of the mine, to avoid unnecessary travelling. The cost of a comfortable stable is about $10 per stall. Special attention must be given to proper ventilation, drainage, and cleanliness. The opening must be not less than 18 feet wide.

Animal haulage is expensive and is fast being supplanted by mechanical systems. The number of working days at a colliery seldom exceeds 200 in a year. The cost of maintaining the animal during the remaining 165 days, besides those resulting from depression in trade, strikes, accidents, and diseases, adds materially to the cost of production. Moreover, horses are subject to epidemics, and it may happen that in times of greatest prosperity some or all of the horses may be unable to leave the stables. Delays arising from the dropping of shoes while at work are costly, for, while the farrier is at work, both horse and driver are standing idle. Out of every thirteen animals in the mine an average of ten may be regarded as a full working

complement, the remainder being on the retired list for various causes.

Compared with the mechanical methods, animal haulage is more expensive. It is slower and therefore used on long hauls only when numerous ventilating-doors intervene along the line.

The Locomotive.—For great distances and a large output the locomotive is much more efficient, more rapid, and cheaper than animal power, being equally flexible and having, moreover, the advantage in that during strikes and lockouts there is no expense in maintaining it. It can be accommodated to varying demands of the different sections of the mine. The motor is in direct charge of the engineer, who is always on hand, so that it can be stopped promptly. This would avert such accidents as are common in the rope-haulage systems. Here the conductor of the train must first signal to the surface before the train can be stopped, and any incumbrances on the track, or the jumping of a car from the track, would result in demolition of the entire train before the rope could be brought to rest.

The locomotive is operated by air, steam, or electricity with nearly equal efficiency and in any desired size for traction purposes. It is difficult to estimate the number in use in the United States, but in Pennsylvania alone are 950 locomotives, 503 being steam-locomotives, 71 operated by compressed air, and 376 electric. The anthracite region uses mostly the steam-locomotive, while the electric motor is more common in the bituminous district.

All traction-locomotives, depending solely upon the adhesion of their drivers to the rails, are limited to grades of 4 per cent. They are not economical on steeper grades, except for short distances and where the up-grade is approached by a down-grade, enabling the train to acquire momentum enough to help it up the steeper grade.

The Haulage Capacity of a Locomotive.—This is measured by the weight of the train which it can haul upon a level straight track. Its tractive force, or its draw-bar pull, is the tension exercised by the locomotive upon its first connection with the train.

The draw-bar pull must exceed the total frictional resistance of the train upon the track, including that between the wheels and the rail, the resistance of the curves and of the grade. The draw-bar pull is about one eighth the weight of the locomotive. Hence the weight of a locomotive should be the maximum for given minimum dimensions.

The Tractive Force.—The tractive force, T, of a steam- or air-locomotive, expressed in pounds, is measured by the formula $dT=12k^2sp$, wherein d is the diameter of the driver in feet, and k, p, and s have values as in Chapter V for steam-engines. The traction of a locomotive, expressed in pounds, must not be confused with its horse-power, which is a unit of dynamic force, embracing the elements of weight, distance, and time.

The train resistances R are: (1) the frictional, which in mines is not less than 30 lbs. per ton of train, and equals $30Y$, Y being the number of tons weight of train and load; (2) that due to grade, which is $20gY$, g being in feet per 100; (3) that due to the curve, which is $\frac{1}{8}$ lb. per ton per foot width of gauge, z, per 1° curvature; and 4, that due to speed, which is $0.25V+3$, V being the speed in miles per hour. For any degree of curvature, D, and weight, w, of cars which are at one time on the curve, the curve resistance is $\frac{1}{8}wzD$. As the radius of a 1° curve is 5730 feet, then, when the radius of the curve, ρ, is known, the curve resistance $=716.2\frac{wz}{\rho}$. Often the equivalent grade resistance is calculated; that is, the percentage of grade which offers the same resistance as the curve. This $=1.32D$.

Example.—A locomotive has a draw bar pull of 2400 lbs. How many tons will it pull up a grade of 1 in 50 if the friction is 32 lbs. per ton? The work against gravity is 1/50, or 45 lbs. per ton, making a total resistance of 77 lbs. The available pull being 2400, the gross load is 31 tons, less the weight of the locomotive.

The influence of grade in reducing the haulage capacity of the locomotive is shown in the following table. For any other weight of locomotive the capacity is directly proportional.

HAULING CAPACITIES OF LOCOMOTIVES ON VARIOUS GRADES, IN TONS OF 2000 LBS.

Weight.	Draw-bar Pull on Level.	Frictional Car Resistance per Ton on Level.	Grades.					
			Level.	½ Per Cent.	1 Per Cent.	1½ Per Cent.	2 Per Cent.	2½ Per Cent.
4,000	500	20	23	15	10	8	6.3	5.2
		30	15	11	8.4	6.7	5.4	4.5
		40	12	9	7.0	5.7	4.7	4.0

Weight.	Draw-bar Pull on Level.	Frictional Car Resistance per Ton on Level.	Grades.					
			Level.	3 Per Cent.	3½ Per Cent.	4 Per Cent.	5 Per Cent.	6 Per Cent.
4,000	500	20	23	4.2	3.5	3.0	2.2	1.6
		30	15	3.8	3.2	2.7	2.0	1.5
		40	12	3.4	3.0	2.5	1.8	1.4

Comparison of Types of Locomotives.—At first the steam-locomotive was used underground, but the objections to it, which were soon recognized, led to its removal from gaseous mines or from those whose ventilating system may become disturbed by the passage of the locomotive through the airway, and it has been supplanted by the later forms of motor-power.

The compressed-air locomotive is the only cheap form of haulage for gaseous mines. It introduces no element of danger other than that common to all large haulage motors—the interference with intake air-current during the outward passage of the locomotive.

The electric locomotive has now established itself in coal-mines, and may be regarded as a safe form of haulage except in special cases. It may be had of three different types—the overhead-wire trolley, the third-rail, or the rack-and-pinion when the grade is excessive for the simple traction-engine.

On the other hand, a very decided objection obtains against the air- and the steam-locomotive, and to a less degree against the electric locomotive, in the area of the haulage-way which they occupy. Unless the latter is excavated to a large area the passage of the locomotive interferes markedly with the air-current. When

FIG. 111.—An Electric Mine Locomotive.

travelling with the current, at least 20 per cent more air enters the mine than when the train is moving out against it. This may be averted by disconnecting the locomotive haulage-way from the main ventilation system of the mine and introducing the inlet current at a point beyond the inside terminus of the locomotive gangway. The cost of these improvements, however, is very large.

The Steam-locomotive has not as ready acceptance in underground work now that it had in former days, because of the objection raised to the presence of the gaseous products which it discharges into the airway. Whether coal-burning or oil-burning, there is given off a considerable percentage of carbonic oxide gas as well as smoke. The former is a menace, and the latter befouls the ventilation besides introducing the risk of fire. Moreover, the great heat and moisture in the exhaust tend to rot the timbers and to soften the roof. These are about the same objections that obtain against any underground steam-engine. In long haulage-ways steam-locomotives can be used only when a special means of ventilation is provided. There are many instances where lives have been lost by the inhalation of the gases from the locomotive and underground engines. The steam-locomotive, however, is less expensive than the air-engine and very much lighter in weight.

The capacity depends on the driver adhesion, and a steam-locomotive of 6″×10″ cylinders will haul 28 tons of train at a rate of twenty miles per day up a grade of 105 feet per mile with 600 lbs. of coal. One of 10″×14″ cylinders has a duty of 46 tons per hour at 28 miles speed on grades of 52 feet per mile. The coal consumption is 1000 lbs. per day.

The locomotives are made of a shape to suit the mine opening, for narrow gauge (36 to 40 inches) rarely over 78 inches high, have four to six wheels (for curves of 50 to 75 feet radius), weigh 4 to 13 tons, and carry 125 to 350 gallons of water. Their cylinders are from 5×10 to 10×14, on 22-inch to 28-inch drivers running over 16- to 28- lb. rails. These engines cost from $2600 to $4000. They have a traction of from 150 to 600 tons on a level.

There is no difference in the price between the wide- and the narrow-gauge locomotive of the same design and size of cylinders. For narrow tunnels locomotives with inside cylinders may be had.

The Compressed-air Locomotive.—This consists of one or two storage tanks, in which is retained compressed air sufficient in amount for the entire run of the trip, and two cylinders with the usual driver connections. When the air-pressure is as high

Fig. 112.—A Mine Locomotive.

as 1000 lbs. per square inch, the high-pressure tank, placed along the side of a larger tank, is quite small, with a reducing-valve between them. The cylinder receives air from the low-pressure tank. As the radius of action depends on the capacity of the tank, the latter is double that calculated to be required for the trip. The cars are hauled to the foot of the slope, or, if the grade is a natural one, all the way to the tipple. The locomotive goes as far into the mine as the height of the seam and the local conditions will admit.

These locomotives are operated precisely like the steam-locomotive, but are of greater weight, with larger bearings, frames, cylinders, wheels, etc., though the general pattern is much the same. The early objection to the air-engines—the fear of explosion—is no longer warranted. Their dimensions are about as follows: 16 to 20 feet in length, 6 to 7 feet in width, and 5 feet in height. The cylinders are not lagged as are steam-cylinders; indeed their exterior surface is corrugated to increase the exposure.

The haulage capacity is calculated in the same manner as for steam-locomotives, the percentage sometimes allowed for adhesion of drivers to the rails being one fourth. Their tractive efforts are calculated in the same way; after ascertaining the mean effective pressure from a given ratio of expansion and from the size of the cylinder, the volume of air required to deliver the given power at a known rate of speed is found. In determining the mean effective pressure, the second table in Chapter X will be found convenient for conditions in which the air is at a pressure of not to exceed 200 lbs. per square inch. Above that the formulæ may be employed for the purpose.

The following table shows the tractive effort of certain air-locomotives.

TRACTIVE EFFORTS OF COMPRESSED-AIR LOCOMOTIVE.

Cylinder.		Diameter of Driver, Inches.	Weight on Driver, Pounds.	Tractive Effort for each 100-lb. Gauge Cylinder Pressure at Various Cut-offs, for Atmospheric Pressures.						
Diameter, Inches.	Stroke, Inches.			7/8 in., 0.98	3/4 in., 0.95	5/8 in., 0.88	1/2 in., 0.80	3/8 in., 0.68	1/4 in., 0.51	1/8 in., 0.31
5	10	24	6,000	1020	990	920	835	710	530	325
6	10	24	8,500	1470	1425	1320	1200	1020	760	445
7	12	26	13,000	2200	2150	1990	1810	1540	1140	700
8	12	26	18,000.	2880	2750	2600	2360	2000	1510	900
9	14	26	25,000	4340	4140	3840	3490	2960	2220	1350
10	14	26	32,000	5280	5150	4740	4310	3660	2630	1670
11	16	28	42,000	6770	6450	5980	5440	4620	3470	2440
12	16	28	52,000	8050	7800	7200	6550	5580	4150	2850

Aside from its low efficiency, the main objection to the compressed-air locomotive is in the great dimension of its tanks. The exhaust is laden with snow if the ratio of expansion has been high, for the air is usually filtered through water previous to compression, to remove the dust. This charges it thoroughly with moisture, which causes the trouble.

Compound pneumatic locomotives are being introduced in order to utilize the expansive effort of the air. They have an advantage over the simple locomotive of lower piston speed for

the same consumption of air. A higher initial pressure is then used.

The cost of a compressed-air plant, with motors, boilers, etc., is $33,000 for a mine having an output of 900 tons per day with a mean haul of 2000 feet. The cost of haulage per ton mined is 12.6 cents.

The storage tanks on the locomotive are charged at either end of the line from stations where large storage tanks are located, which latter are in pipe communication with the air-compressor. The capacity of these stationary tanks exceeds twice the capacity of the locomotives comprising the system.

FIG. 113.—A Pneumatic Locomotive.

The Electric Locomotive.—This is the most advantageous form of electric application for underground work both in point of economy and convenience. Its introduction has been of slow growth in coal-mines because of the conservatism enforced upon engineers by adverse legislation, which has always compelled them to go slow in the introduction of any new form of machinery. Since its first installation it has established itself and is more popular than the compressed-air motor, because it is more compact and more economical. Its compactness enables it to serve in entries too small for compressed air by engines and too sinuous for cable-ways. The results from its introduction have not proved any more destructive or injurious to life than from other forms of haulage power, and it is used in any gangway where the naked lamp can be carried. It possesses the same advantage as other locomotives in having a practically unlimited radius of action, and has an additional advantage in being cheaper to operate than its

competitive traction machinery, having nothing to get out of order. Because of this little trouble ensues.

Two systems of electric locomotives are used; one with overhead-trolley wire and the other with the third rail.

The mechanism of the underground locomotive is similar in kind to that of the familiar street-car trolley in which hard-drawn copper wire is laid along the roof of the road. This form of the overhead wire on which the trolley of the locomotive arm rests conducts the current to the motor, whence it may return by the rail to the generator.

The third-rail locomotive has a rack between the two main rails in which engage two sprocket-wheels driven by the electric motor by means of suitable gearing. These sprocket-wheels serve the double purpose of driving the locomotive along the track and taking the electric current from the third rail to feed the electric motor. These machines are of recent introduction.

Locomotive Details.—It is impossible to indicate the average dimensions of the locomotive, because each one represents a special type designed for local conditions only. Any desired height, power, or gauge may be procured. Its height is rarely more than 4 inches above the tops of the wheels; its length is 11 feet, and width 57 inches, for a gauge of 36 inches. Its average weight is 10 tons. This would have 50 horse-power capacity and would be capable of taking 25 cars of 4.5 tons gross weight each on the level.

The driving motors are made as small as is consistent with good design, and are suspended as low as safety warrants in the space between the wheels. They combine both high power and good speed; their mechanism is of simple character and easily accessible. A housing affords protection from dust or injury. They are enclosed in a heavy cast-iron frame firmly bolted and carrying axle-boxes. To each axle a steel-clad motor is geared. The gears are of the best cast steel and designed for maximum strains which are sure to occur when, in the darkness of the mines, it may become necessary to reverse from full speed.

Sand is carried in boxes, fore and aft, fitted with spouts in front of each wheel; the brake is designed to be applied by one hand. The screw-brake is preferred to gear or ratchet.

The wheels are generally of chilled iron, 30 inches in diameter, and the trucks of the prevailing gauge. Steel tires are more expensive, but furnish a greater tractive effort than chilled wheels. The length of wheel-base is governed by the existing curves. This is 5 feet for a gauge of 36 inches and a radius of 25 feet.

The trolley-pole is reversible and adjustable to maximum as well as minimum heights. It may be placed on either side or end of the car as conditions require, the centre of the socket being over the gauge line and on the ditch side, to reduce the element of danger. The motor is of the slow-speed four-pole type with windings suited to the given line pressure and a speed of eight miles per hour, taking into consideration the line loss. If it happens that the line loss is less than was assumed, the locomotive speed will exceed this.

The Rheostat.—*The controlling device* is of the five-point rheostatic type of electric railway motor, so designed as not to heat to the danger-point when using the maximum current. This form of diverter, or rheostat, is necessary because of the great variation in load and grade which the locomotive encounters in a run.

The resistance-box is placed in front, where it will receive a strong circulating current of air. Powerful magnetic blowouts are always provided to insure against sparking.

The Draw-bar Pull of an Electric Locomotive. — This is about 50 lbs. per horse-power of the motor, and the weight of the locomotive at the ordinary speed of running is about 400 lbs. for each horse-power of motor, making the tractive effort of an electric locomotive about one eighth of its weight. For continuous work the effective tractive effort should be assumed as 20 per cent less than this rated capacity. This will prevent overheating of the wires.

The average bar pull, P, of different weights, W, of loco-

motives expressed in pounds on varying grades, g, is expressed as follows:

$P = W\ (0.125 - g) =$ horse-power of locomotive $\times (50 - 4g)$. A ten-ton locomotive can pull 1900 lbs. on a 3 per cent grade or 1300 lbs. on a 6 per cent grade, as against 2500 lbs. on a dead level. The starting draw-bar pull is nearly 60 per cent greater than the ordinary running rate when sand is used.

Locomotive Rating.—The length of time for which a motor can give a stated ampere rating, I, is based on the expression tI^2,

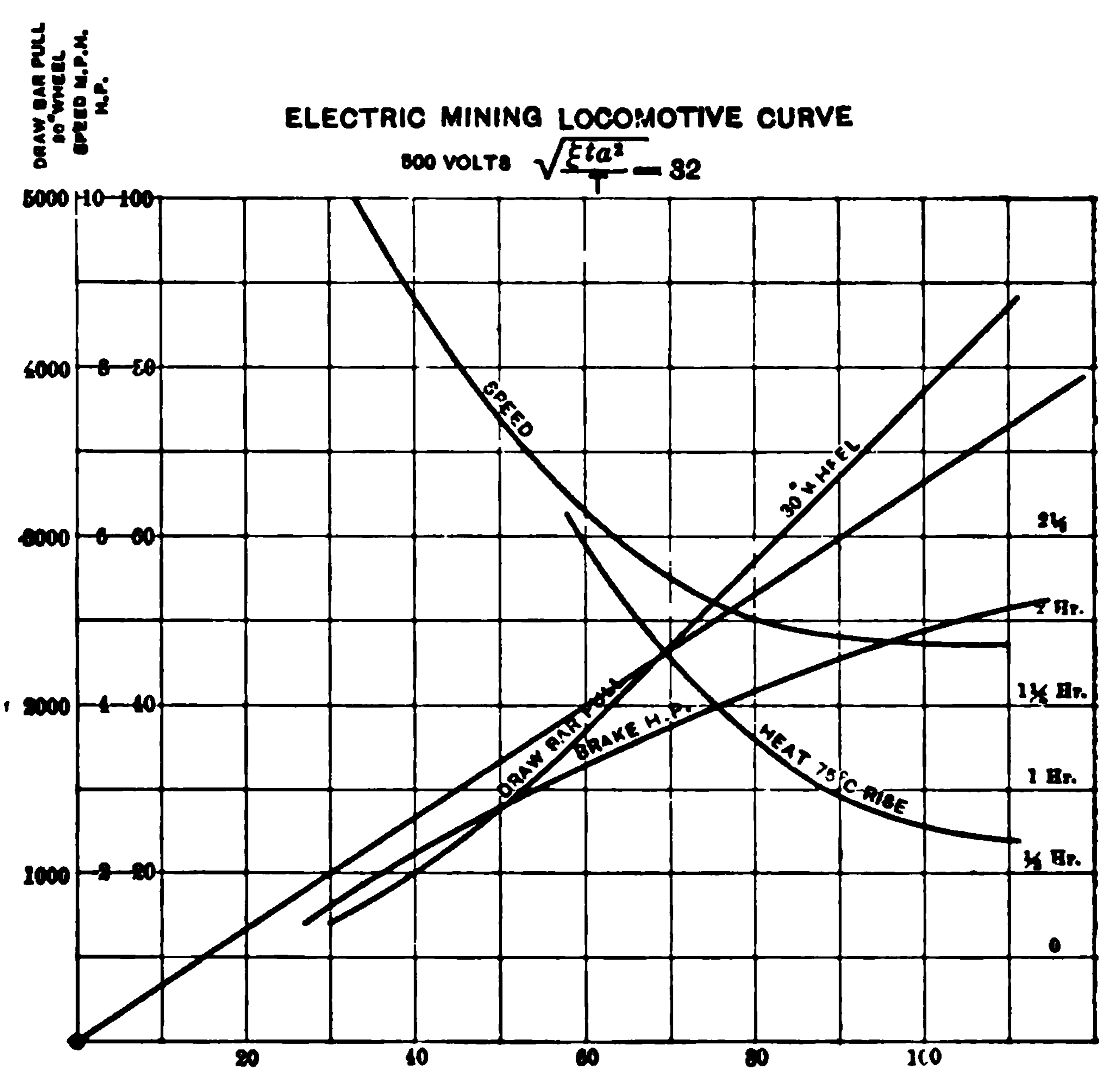

FIG. 114.—The Power Curves of a Locomotive.

in which t is the number of minutes of running, and I the number of amperes for regular work. Every overload causes heating beyond the allowed limit. This varies with the square of the current. The rating of a given locomotive being known, it is possible thus to ascertain its hauling capacity.

Thus a motor rated at 50 amperes for 18 minutes can work only 4½ minutes at 100 amperes. The length of haul also has an influence on the capacity of the motor. One intended to pull a maximum load for a long period must have a greater capacity than one which is operating for short hauls, allowing time to cool between stops.

Notation.—A system has been adopted for notation of electric locomotives similar to the Whyte system for steam-locomotives. The number of wheels under the locomotive, the number of motors and horse-power of each, and the number of pairs of drivers geared to the motor are indicated in the order named. The letter indicates the number of drivers: *B*, for one-geared axle; *C*, two pairs of drivers; *D*, three axles geared to motors. $4\frac{2}{50}C$ means 4 wheels, 2 motors of 50 H.P. each, and 2 geared axles.

Storage-battery Locomotives.—Owing to the numerous accidents from contact with the wires, whereby the mule becomes an unnecessary evil in the operation of the mine, the animal may be eliminated from the haulage systems by employing manual labor for pushing the car from the parting into the entry. A combined storage and trolley locomotive has also been devised for the purpose and employed with success. Storage batteries obviate the necessity for wires, but as yet are too expensive and also too heavy for efficient use. A 25-lb. battery will furnish one horse-power hour; and a space of 40 sq. ft. will accommodate 250 elements, which would be able to give 11 horse-power during a full ten-hour shift. This weight and the adhesion is more than twice that of the steam-locomotive of equal horse-power, but requires a heavier rail.

EXAMPLE.—It is desired to move sixty cars of a capacity of 3 tons each and weighing 1 ton over a track having the following profile, favoring the loaded cars: 1920 feet of 1.3 per cent grade, 1000 feet of 1.8 per cent grade, 600 feet of 2 per cent grade, 400 feet of 2.5 per cent grade, 425 feet of 3.5 per cent grade, and 1000 feet of 0.9 per cent grade. The locomotive rating is 64, voltage 500, amperes 32 for each motor, and speed 6 miles an hour.

The maximum resistance is on the steepest grade, or 4½ per cent.

If the car friction be assumed at 1 per cent, the cars will require 5400 lbs

draw-bar pull to do this work. To provide for the friction of the locomotive itself, which is 1280 lbs., the total tractive effort must be 6680 lbs. With cast-iron chilled wheels the weight of the locomotive must be 53,280 lbs. Dividing the draw-bar pull by 50 shows that the locomotive must be equipped with 108-horse-power motors to do the work without overload. For continuous work we have, for the two motors,

$$\sqrt{\frac{\Sigma tI^2}{T}}=2\times 32.$$

T = the total time that should be taken for each trip. Calculating for the several sections the time of haul in minutes and the required amperes, respectively, as below, with a voltage of 500:

Time	=	3.8	2	1.1	0.8	0.8	2 minutes
Amperes	=	114	128	134	148	182	100
and tI^2	=	41,400	32,768	19,751	17,523	26,500	20,000

we have $tI^2 = 165,942$; $(64)^2T = 165,942$; and $T = 40$.

Cost of Electric Haulage.—The life of the average electric-haulage plant is estimated as twenty years, and hence in distributing the cost of haulage an allowance of 5 per cent depreciation per year divided by the number of working days per year will give the allowance to be made for this item per day.

Comparing some instances of the cost of electric haulage with that of mules, the evidence preponderates in favor of the former. On the item of track cost alone it shows a saving of 25 per cent on the investment per year.

Self-acting Planes.—Conditions occasionally occur when gravitation can be utilized to deliver the loaded car to the bottom of the plane. A double line of rails is laid the entire length of the track for two trains of cars. A rope connects them and passes around the sheave at the top of the incline fitted in a recess above the landing where the cars are started. In some cases a single rope is used, the sheave being horizontal and its axis vertical; in other cases a drum, whose axis is horizontal, carries two ropes wound in opposite directions, each one having one or more cars attached to it when desired. A single rope may be used on the drum with five or six extra coils upon the drum between the two branches of rope to give friction enough to prevent the rope from slipping. This form of plane is used in haulage-ways,

and occasionally in individual rooms for the delivery of one or more cars at a time.

Sometimes three rails are laid with a suitable turnout midway along the incline. This, however, is false economy, as compared with the full four-rail line.

The lowest gradient at which the plane is self-acting depends on the state of the road and the load, but under the most favorable conditions the gradient requires to be about 1.25 inches per yard, or 1 in 29. At about 6 per cent pitch it works freely. The most satisfactory grade is 1 in 6. When the grade is as large as 1 in 3 there is an excess of motor force above the resistance, and a suitable brake must be supplied to control the speed. Some inclines have a counterbalance-car. The average safe velocity on such a plane is 400 feet per minute. A two-car plane on 10° incline requires a $\frac{5}{8}$-inch rope. On a 45° slope the rope is $\frac{7}{8}$ inch in diameter for three cars.

The lower portion of the inclination is flattened somewhat to diminish the momentum of the descending cars, and in some cases a reverse grade is built in order to check the train. The speed of the car or train is regulated by a brake applied on the band bolted over the sheave-wheel. It is of the usual type, fitted with good durable surface having a high coefficient of friction. The safety attachments provided on cars are essentially confined to some form of trailing-block.

Single-track self-acting inclines for single cars have a trailing counterbalance-car to regulate the speed, with a turnout at the middle of the line.

Engine Planes.—When the slope is against the load and a locomotive cannot be employed to advantage, engine planes are used. A stationary engine with drum is located at the head of the plane to raise the cars in the train at any speed which the timbering of the roadway will permit. If the traffic is large enough to warrant it, the engine is in continual motion, the drum being thrown in or out of gear when desired. If the drum may turn freely, it will pay out the rope for the descending cars, and is geared to pull them up. On a grade of 1.7 per 100, gravity will

FIG. 115.—A Self-acting Inclined Plane.

take the loaded cars down with a reasonable velocity (empties on a 2.25 grade), pulling the rope behind them. On a 10 per cent grade a brake will be necessary for the empty cars.

The track may be single or double, according to the output desired. On single-track planes the engine is non-reversing. The cars usually travel in trains, ten to thirty cars to a train, each in charge of a conductor, who operates a dead-fall timber block to hold the train while the cars are being shunted. The size of the engine can be calculated as for hoisting, by determining the amount of power requisite to overcome the vertical component of the weight of the rope and train and the horizontal resistance of the friction. The engine plane is eminently suited to an oil-engine or an electric installation. It is well adapted for the delivery from side entries at different levels, and may be used on slight curves by curving the rope on iron guide-wheels. Ordinarily the rope will last four years. A $14'' \times 30''$ engine with a 3-ton fly-wheel, $\frac{9}{16}$-inch steel rope on a plane 4600 feet long and 80 feet rise, has a daily output of 950 tons of mineral in trains of 25 to 30 cars.

Rope-haulage Systems.—The two classes of rope-haulage systems are known as the tail-rope and the endless rope. Both are extensively used in horizontal seams, and while each has its special adaptation to a given grade, local conditions may make it possible of application under steeper grades. The tail-rope can be employed for grades not exceeding 3 in 100, either with or against the loaded cars.

The Tail-rope System.—This consists of a stationary engine with two drums which are thrown alternately in and out of gear. From each drum is carried a rope. The main-rope has a length equal to that of the road, and is fastened to the front end of the train. The tail-rope, of double this length, passes from the drum around a sheave at the end of the road and up to the rear end of the train.

When hauling the loaded train out, the main-rope drum is thrown into gear and that carrying the tail-rope is free. The loaded cars are then driven to the outlet or the point of distribu-

FIG. 116.—A Tail-rope Haulage Engine.

tion, dragging behind them the tail-rope. When the train has been replaced by empty cars, the main-drum is released and the tail-drum engaged. The engine is started and the cars are delivered into the mine. The lower sheave is a grip-pulley similar to that used in the power transmission, Fig. 118. The main-rope is rarely over $\frac{7}{8}$ inch in diameter. The tail-rope is usually an old discarded main-rope. Swivel connections are used from the rope to the train.

The roadway is usually a single track, and the sheave is vertical with the tail-rope carried along the roof, while the main-rope is supported along the floor. The length of haul is limited only by the engine power and the track resistances. The most advantageous pitch is about 3 per cent outward from the workings. The velocity of haul may reach as high as 10 miles an hour. Each trip may take a train of from 10 to 50 cars, usually accompanied by one trailer in charge of a conductor.

This system is very much used in American collieries. It is the best plan for operating branch ways. Each branch has its own rope passing over sheaves at the ends. When a train is to be hauled out of the room to the point of discharge, the principal ropes are opened at the proper points and connections with the branch ropes are made; meanwhile the other branches are idle. This is an inexpensive plant to build and to maintain. It is advantageous for undulating roadways and has very largely replaced animal power.

An engine of $18'' \times 30''$ cylinder with 75 lbs. steam-pressure draws thirty trains of 17 cars each in a day over a slope of 2800 feet long with a grade of 1 in 200. The drums are 4.5 feet in diameter, the main-rope is $\frac{3}{4}$ inch and the tail rope $\frac{1}{2}$ inch diameter. The velocity of travel varies between 8 and 11 feet per second.

A tail-rope plant for 1000 tons daily for a mile of haulage will require about 130-H.P. boiler and engines; 6000 feet of main-rope 1 inch diameter; 16,000 feet of $\frac{3}{4}$-inch tail-rope; 600 rollers; 500 cast-iron sheaves; 10 return sheaves and drums. The first cost would be nearly $7500, and the maintenance about $20 per day.

The Endless-rope System.—This system is much in vogue. A rope or a chain receives continuous motion in one direction from a wheel or drum, and draws the cars out on one line and in on the other. The tension requisite to carry the load is artificially produced by a carriage provided with some form of counterpoise. It requires a double-track line for communication and considerably less rope than in the tail-rope system. Cars can be attached at any point on the line, but branch connections are not possible with this system.

The wire rope or chain cable may rest on the top of the car or be supported below it. In the cheapest and most universal method the chain rests in forks riveted on the top of each car. For uniform grades and sharp curves the endless rope may be suspended above the cars, to which it is attached by short lengths of chain. If the chain cable is underneath the cars, it runs on rollers, connections being made by hooking a short length of chain from each car. With a rope running over rollers or sheaves on the floor, like the surface cable roads, connection is made by a hand-grip attached to the car.

The engine of the double-track system is in continual rotation, driving a shaft fitted with a heavy fly-wheel, and with a friction-clutch for emergencies. From the shaft a belt or rope drives a parallel shaft with the sheave about which the endless rope travels. A similar sheave at the far end of the line, with a balance-car or tightener, constitutes the entire mechanism. The speed of the rope rarely exceeds four miles per hour.

The cars are attached singly at the landings as fast as they arrive. The connection is made by wrapping a few turns of chain about the rope and hooking the ends on by the hand-grip. The latter is more convenient and secure than the former, particularly if the cars are in train. The grip is a pair of clamps attached to one end of the car, the end of the movable-jaw lever being closed on the rope by a ring slipping on to its mate. Raising the ring opens the clamp and drops the rope. The car is released and switched to its proper track. In the same way it can be as easily clamped on the rope to pull the car. For a train of cars a leading grip-car is attached by a gripman, who

takes the train out or in. Clevis couplings are also used for single cars and at the front and rear ends of trains. Sometimes a "knock-off" hook is used at the rear to aid in uncoupling while strain is on the rope.

Increasing the Tension of the Drums at the Driving End.— The difference in tensions due to the two loads pulling upon the two branches of rope is slight. The amount of friction at the sheave must at least equal the difference of loads. Should any excessive loading or resistance occur, the sheave will slip and fail to drive. A tighter grip must be taken on the rope by the sheave to communicate power. It would also prevent excessive wear of the rope. This is possible where the sheave has several grooves. The number of coils will be such as will provide the necessary friction to prevent the rope from slipping off. This may be determined by the formulæ for power transmission. Two drums in tandem serve much better, each one having four or five grooves, the rope making a half-coil alternately upon each drum in each groove before returning along the other trackway. They are both geared to the engine for equal speed.

The Balance-car.—The variations of load and changes of temperature produce varying tensions and elongations of the rope. To insure a uniform tension the sheave at the other end of the line is mounted on a carriage and capable of sliding freely. Beyond it is a sunken pulley over which is the rope from the carriage to a suspended counterpoise. When the loads on the rope become heavy, the carriage is pulled by the pendent weight to increase the rope tension. With a light load the weight of the rope draws the carriage. If the grades favor the load, the carriage is at the end of the run of the loaded cars, the power being inside of the mine at the top of the empty track. On aerial tramways the same regulating device is used, and in Fig. 118 is an illustration of one.

Instead of the plain grooved sheave there may be a grip-pulley, several feet in diameter, carrying radial forked arms to seize the rope. The arms are capable of being lengthened or shortened as the rope stretches and drags. When the arms have

been extended to their limit, a short length of rope or a few links of cable are removed and the forks readjusted.

An 18"×42" Corliss engine, with two 36-inch driving-pulleys, 18-inch belts to a pair of 84-inch pulleys, and two pinions of 18 inches diameter with 26 teeth, gearing into 72-inch spur-wheels with 91 teeth, drives two 72-inch rope-drums. The drums with four grooves each operate an endless-rope system one and a half miles long, delivering 1200 tons daily over an average road. The fuel, labor, etc., of a 450-ton endless ropeway is about $7.50 per day. The initial cost is about $6700 for a plant of 4200 feet double track (1500 feet having a rise of 40 feet, the rest being level). The rope is $\frac{7}{8}$ inch diameter.

If only a single track is possible in the haulage-way, the operation is intermittent. The engine alternately pulls the rope with its train inward and outward, being reversed for each trip. This resembles the tail-rope system with the two ropes linked together. One line of rope is supported on overhead sheaves, and the other, to which the cars are attached, rests on the floor rollers. The cars are in train.

The road may be more readily extended with this system than with the tail-rope. It costs less for power and can be carried around sharper curves and over steeper grades. It can be used with practically every condition of roadway with greater economy than any of the other systems. The delivery of cars is uniform and continuous, regardless of the length of haulage. The amount of car and rope repairs is less than in the tail-rope system. The risk of accident to men along the roadways or damage to property is also less, because of the slow speed. Though applicable to sinuous undulating roads, it is best adapted to uniform grades throughout the entire length of the roadway, because of the varying load which would otherwise be thrust upon the engine where the grades are steep and variable. This would be particularly true when the intermittent attaching of cars is very irregular.

Supporting the Haulage-rope.—Rollers and sheaves along the haulage-way support the rope to reduce its wear and to dimin-

ish the frictional resistance. They are placed midway between the rails on suitable blocks spiked to the ties, and 20 to 30 feet apart. They are also employed for supporting the tail-rope near the roof by being suspended from the roof-timbers. The rollers are usually of wood, 12 to 18 inches long, 5 inches in diameter, and carefully turned with an iron journal for support. They are spaced on a level track from 15 to 20 feet apart (closer together if the rope is large or the grade steep). The bearings also are of oak, well oiled. Hollow rollers may be obtained of cast iron with curved faces, to keep the rope as central as possible. They are shorter than the wooden rollers and offer much less resistance.

Deflecting the Main-rope Around the Curves.—This is accomplished by a number of small sheaves laid on the centre line of the track, concentric with the rails. They are of a diameter of 10 to 15 inches and are spaced 3 feet or so apart to give a slight deflection of each one for the entire length of the curve. On reverse curves they are laid nearer the inside rail than the outer rail, with their axis somewhat inclined and leading outward. The formulæ for determining their spacing are to be found on page 177.

Short vertical rollers are also placed on the ties when the rope is to be deflected around a gentle curve. For a sharp curve the rollers are frequently placed at the side of the track, with their axes slightly inclined from the vertical to carry the rope. These are spaced 3 feet apart outside of the inner rail of the curve. They are usually of cast iron, chilled, with an upper flange. Guide-blocks are laid at intervals from the inner rail toward the sheave on all simple curves in order to guide the rope into its position in the groove, and also to prevent its catching in the rail as the train moves around the curve. These blocks are of oak, spiked to the ties.

Supporting the Tail-rope.—Tail-rope sheaves are hung on the posts at the sides of the haulage-way, 25 to 30 feet apart. For these a perfect alignment is necessary, in order to reduce friction and increase the security to the rope in the groove. They

are of chilled cast iron or case-hardened, having wooden filling in the grooves, and are from 10 to 16 inches in diameter.

The tail-rope is turned out of a side heading by a horizontal guide-sheave of a diameter as large as convenient. Its pattern is like that of the clip-pulley described under Power Transmission, Chapter IX. A large diameter is necessary because the angle of contact exceeds 90°.

An electric system along the tramways is requisite for safety as well as for signalling, though the malicious destruction of insulation, etc., has caused its abandonment by many operators.

EXAMPLES.—1. A one-mile endless rope, travelling at two miles an hour over an average grade of 3 in 100, delivers 50 tons per hour. Required the total resistance on the line and the size of the cylinders under 50 lbs. boiler pressure and 160 feet piston speed. Each car weighs 800 lbs. and carries 2000 lbs. Along the line are distributed 25 loaded and 25 empty cars, about 100 feet apart, and a ton is delivered every 36 seconds; with a coefficient of friction of 0.02, the frictional resistances on the halves of the line are 1400 and 400 lbs. respectively. To raise 25 tons up the plane requires a force of 3 per cent of 50,000, or 1500 lbs.; the gravity component of the cars is 600 lbs. The gravity components of the cars and the rope, up and down, balance each other, leaving the work of the engine to be that of overcoming 1500+1800 (the drag of the rope is assumed at 3000), which, carried at a rate of 176 feet per minute, requires 1,109,800 ft.-lbs., or 33.6 I.H.P. The piston diameter, k, is $7\frac{1}{2}$ inches, and for 130 strokes, s, nearly 15 inches. If the driving-sheave is 6 feet diameter, it makes 9.3 revolutions per minute, and is therefore geared 1 to 7.

2. A haulage-engine having cylinders 24″×48″, at the head of a plane of 10 per cent grade, is directly connected to its drum. With an effective steam pressure of 45 lbs. per square inch, a piston speed of 350 feet per minute, and a train speed of 8 miles an hour, what is the size of the drum and the capacity of the plant?

The total piston pressure is 27,150 lbs., assuming a modulus of $\frac{2}{3}$, which, at 350 feet per minute and a train speed of 704 feet per minute, represents a continuous load of 13,497 lbs. This tension upon the hauling-rope, due to the component, parallel to the plane, of the weight of the rope, cars, and load, plus their friction, is therefore limited to 13,497 lbs. For this working load the rope may be $1\frac{1}{8}$ inches diameter, weighing 2 lbs. per running foot; the cars may be assumed to weigh one half that of their contents, W, and the frictional resistances may be allowed for at the rate of 50 lbs. per ton of normal pressure. Then, by formulæ in Chapter V,

$$13,497=\frac{10}{100}(12,000+\tfrac{3}{2}W)(1+0.025).$$

W becomes 79,840 lbs., and the hourly capacity less than 160 tons, without allowing for delays.

3. To ascertain if the engine is properly proportioned for starting the given load, we examine the table on page 131. Note that the coefficient is 0.3974, which multiplied by $45 \times (24)^2 \times 4$ gives a minimum moment of 41,202 ft.-lbs. With a direct-acting engine the drum must have a diameter of 5 feet 6 inches. The engine can therefore start from the bottom a load, W, greater than 99,300 lbs.

4. If the plane were double-tracked and the engine supplied with two drums fast on the same shaft, the only work falling upon the engine would be that due to the contents of the cars plus the total friction on the double line. In this case seven full-length trips might be made per hour, thus increasing the hourly capacity to 377 tons.

5. Required the length of a haulage plant for a tail-rope system to deliver 1000 tons in ten hours over a road the first 600 feet of which has a dip of 4 feet per 100 with the load, the next 2100 feet a grade of 3 feet per 100 against the load, and the lower 600 feet 2 feet per 100 against the load.

Assume that each car weighs 1000 lbs., and that its capacity is 2000 lbs.; assume also a piston speed of 200 feet per minute and an average train-speed of 6 miles an hour.

On the lowest section gravity produces a tension on the main-rope of 5665 lbs., and the engine has to perform 2,991,120 ft.-lbs.; on the next upper section for 4 minutes 122.6 horse-power is necessary to pull the train up-grade. At the head of the gangway the grade favors the loaded cars, which produce a tension on the tail rope of over 6000 lbs. With a drum of 132 inches diameter, making 15.3 revo utions per minute, gearing to a pinion of 0.40 its diameter, a modulus of $\frac{3}{4}$, and an effective steam pressure of 40 lbs., the diameter of each cylinder would be 21 inches and the stroke 30 inches.

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CHAPTER IX.

WIRE-ROPE TRANSMISSION.

Ropes for Power Transmission are of different materials, cotton, hemp, manilla, and steel wire being the most common. For mining purposes the cotton ropes are relatively weak compared with manilla, and have given place to the latter, which are still largely employed for some purposes of driving. If care is taken in the proper selection of a vegetable-fibre rope and in providing it with large pulleys, its durability is increased to an extent that will make it a good competitor of wire rope, which is the most extensively used for power transmission. The steel-wire rope is of the 7-wire, 6-strand type of a diameter of $\frac{3}{8}$ to $\frac{7}{8}$ inch, according to the amount of power desired.

Rope-driving Systems.—Motion may be imparted to an endless rope by its adhesion to a sheave around which it passes and thence to a distant sheave. If the distance between the sheaves is great, the rope is supported at intermediate points. The amount of power which can be delivered to the far sheave depends solely upon the amount of adhesion between the rope and the surface of the sheave. In the American system one endless rope is wound several times around the driving and the driven pulleys. In the English system the sheaves have multiple grooves. Several independent ropes run side by side, each driving in its own groove. The latter, or multiple, system more easily transfers power between the generator and motor, and is more secure against the possibility of breakdowns. On the other hand, the ropes have an unequal stretch and are liable to drive at different velocities and tensions, thus introducing loss

of power, greater wear, etc. In the American, or continuous, system there is only one splice in the rope and no differential driving is likely to exist.

The Transmission of Power.—The amount of adhesion determining the power transmission depends on the friction between the surfaces in contact, the angle of wrap, and tensions on the rope. Being of slow speed, the tensions in the two branches of the rope are due solely to, and depend directly upon, the stretch produced by the weight of the sections between the supports and the amount of sag.

Sheaves and End Carriages.—The sheaves around which the power-rope travels are horizontal, the upper (Fig. 94) being on a fixed frame, and the lower one at the end, remote from the power, on a carriage-tower frame. These sheaves are of large diameter keyed to a heavy shaft, the latter running in a step at its foot and held in a box at the upper end. The rate of revolution of the sheaves is 125, or less, per minute, the speed of the rope being maintained below 4200 feet per minute. The rim of the pulley holds in its groove a number of wooden blocks, or a strip of rubber, to furnish as large a friction as possible. In some cases the rim is composed of a number of gripping-jaws, which are capable of locking on the rope. The brake-wheel is bolted above the arms of the grip-sheave, and is controlled by an adjusting-screw and hand-wheel. In power transmission both frames carrying the end sheaves are fixed, and provision is made for the stretch of the rope by a sheave with arms which are capable of extension, thus enlarging the diameter of the sheave. The rope may also be cut out and respliced.

At intermediate points on the line sheaves are supported on the pillars or towers to relieve the rope of the excessive stress of a long rope. These are of cast iron, 18 or 20 inches in diameter, and fixed on the ends of cross-arms. They run loose on their axles. The grooves are deep and wide to admit the rope nearly to the bottom, the sides being curved to let the rope slip easily. Occasionally the grooves are lined. The distance between the sheaves is governed by the allowable tension and sag.

For power purposes the maximum is 450 feet; for aerial tramways it may be only 200 feet. If they merely support the ropes, they are small in diameter, but if intended to deflect the rope, their diameter is increased. Care must be taken in their alignment in a vertical plane and that the axles each have as little friction as possible. On steep inclinations a second sheave vertically above the lower supporting sheave is placed to prevent the rope from flying from the wheel. Where it is possible for power transmission, the upper branch of the rope is made the driving portion, and the lower branch its slack part.

Maintaining Uniform Tension.—All single-rope power-transmission lines require some form of tightener to maintain a constant tension upon the rope and a uniform driving power. Wire-rope lines for endless-rope haulage and aerial tramways are illustrations of this type. In the English multiple-rope system tighteners are not used.

The tightener may be a sheave or a pair of drums on a movable carriage. When the drums are at the power end of the line they are placed on the same base with their axes parallel. The rope is wound consecutively from one to the other of the drums and around all of the grooves. Either or both may then act as the driver if they are geared together. The two may also be mounted on a carriage, in which case they are at the driven end of the line. The carriage is free to travel upon a track and is fitted with a heavy pendent weight, such that when the rope stretches the counterpoise will pull the carriage to balance the tension, as in Fig. 93. The length of the travel of the carriage must be sufficient to allow of a freedom of movement within limits of variation in rope length caused by the changes of load.

The Tension of the Rope.—On the tight side the tension is that which is due to the centrifugal action of the pulley-sheave, the bending stress of the rope, and the power which the latter receives from, or remits to, the respective sheave. This must never exceed the maximum safe stress of the rope and should be less than one third of the ultimate strength. The tension on the slack side is equal to that on the driving side less the power

taken off by the sheave. It is from a half to one quarter of the tension on the driving side, and the relation between these three forces is indicated by the following formulæ:

Let v = velocity in feet per second;
d = diameter of the rope, inches;
R = weight of the rope per foot;
L = distance between stations, feet;
W = weight on the journals;
n = revolutions per minute;
S = maximum safe stress of the rope;
k = bending stress of the rope;
T_1 = available working tension of the rope, maximum;
C = stress due to centrifugal force;
T = tension on the slack side of the rope;
P = power transmitted to the pulley, lbs.; and
H = horse-power transmitted.

Then

$$C = 0.13v^2; \quad T_1 = S - C - k; \quad P = T_1 - T; \quad \text{and} \quad H = 0.00182Pv.$$

The influence of centrifugal force is negligible in power transmissions for underground haulage or aerial tramways. In these, too, the difference in tensions is not great, for the dead load of cars, buckets, or conveyors is exerted nearly equally upon both branches of the line.

The Sag of a Rope.—The tension upon the rope may be regulated by the amount of sag, or deflection, permitted between the two points of support. The relation between them is indicated by the formula below. Having calculated the sag for a given power condition, a mean between h and h_1 is taken. The end sheaves are separated sufficiently to reduce the sag at rest to this amount. When in action the desired tensions will be obtained. If more power is desired, the sheaves are separated somewhat. If the desired amount cannot be obtained, the rope is cut and spliced. This must be done every three months, as wear and use weaken the rope.

The deflection of a power-rope at the centre of its span in feet being h for the slack and h_1 for the tight portion, then

$$h_1=\frac{RL^2}{8T_1} \quad \text{and} \quad h=\frac{RL^2}{8T}.$$

Where possible, the upper branch of the rope should be the driver and the lower branch the slack portion. An average sag on spans of 150 to 250 feet is about 3 feet when the rope is at rest, this amount being about one half of the sag permitted for the slack portion when in action.

EXAMPLE.—Power is to be transmitted by a steel 7-wire rope, 1 inch in diameter, at a velocity of 4200 feet per minute over 12-ft. wooden grooved pulleys 400 feet apart. Required the horse-power which can be transmitted and the deflections to be given the rope.

Let $S=21{,}000$ lbs. for the rope, and assume the tension on the slack side to be one half that on the tight side.

Here $v=70$; $T=0.5T_1$; and $R=1.58$.

Then $C=0.13v^2=637$; $k=10{,}073$ lbs.; $T_1=10{,}290$; $T=0.5T_1=5072$; $P=5072$ lbs.; $H=646.47$ horse-power as a maximum; $h_1=8.2$ feet; and $h=4.1$ feet.

Aerial Tramways.—Elevated endless ropes are a special form of power transmission. They have long been used as a cheap mode of transportation, serving as feeders to establish systems of communication. They consist of a ropeway elevated above the surface, with carriers for mineral swinging free above the ground. The elevated structure is essentially a series of pillars, or towers, of a strength depending upon the load and its speed and the inclination. It is employed where the contour is too rugged for economical road-building, and is therefore attractive to the miner and the quarryman. The mineral is charged into the carriers at the mine and these are allowed to descend, taking with them the rope to which they are attached if the grade is sufficient, or are pulled downward if power must be employed. A succession of carriers is loaded at the top and delivers the contents at the bottom, where they are automatically emptied without stop, and thence returned on the empty line. The rope

may be straight and of uniform grade, or follow the surface configuration over undulating profile and around curves for distances of a mile or more, provided the power be sufficient to overcome the resistance.

The power required to operate aerial roadways depends upon the grade and average inclination of the line. In the simplest case the grade may be uniform to the point of shipment. Then the inclination alone will determine if auxiliary power is necessary either to assist the load or to check its speed. If the inclination exceeds 20 per cent, the system is self-acting and only a slight braking power is requisite at the top of the line. If it exceeds this amount, power is required to check the speed. The brake must overcome the excess of the downward effort of the load over the resistance of the rising empty bucket and the total friction. If the grade is below 20 per cent, power may be required to supplement the force of gravity. It is not necessary that the grade be uniform, provided there is ample fall from the mine to the roadway. In this event great care must be exercised in determining the value of the motor force as well as all the elements which contribute to the resistance. If power is required in such a case, it may be applied at either end of the line upon a grip-sheave. The motor attached or belted to the driving-sheave must be of the slow-speed type. The superstructure is light and inexpensive. There is no interference with or from surface travel. Climatic conditions do not interrupt operations. The time for their construction is comparatively short, and they have a capacity equal to that of the average mine usually located in such districts. There is no machinery along the line except the towers with their sheaves. In elevated ropeways only such driving and controlling apparatus is built as may be required for maintaining a constant speed and uniform tension. The expense of operation is comparatively low.

The Single-rope Tramway.—The earliest of these tramways was a single endless moving rope, of which the Hodgson and Hallidie are the representatives. The buckets or the carriages are attached to saddles which ride on the rope. They are either

attached permanently to the rope or may be adjustable. In both these patterns one and the same rope moves the load (Fig. 117).

Fig. 117.—A Single-rope Aerial Tramway.

The Ore-carriers.—The mineral is transported in buckets of various designs, according to the character of the material to be handled. They are suspended by hangers or clips, which are either inserted into the rope or clinched around the outside of it, and attached at intervals determined by the amount of material

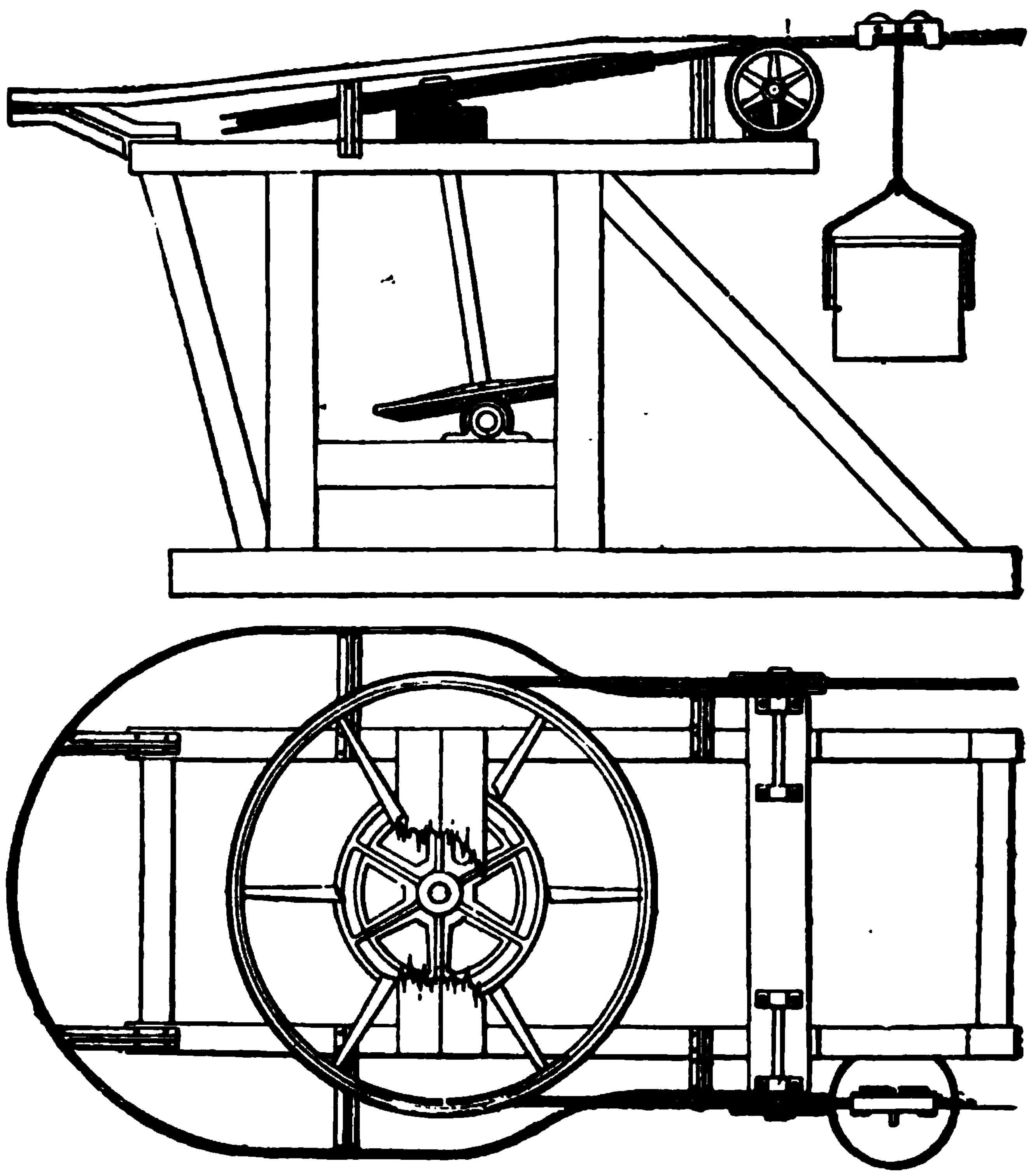

FIG. 118.—The Power End of the Hallidie System.

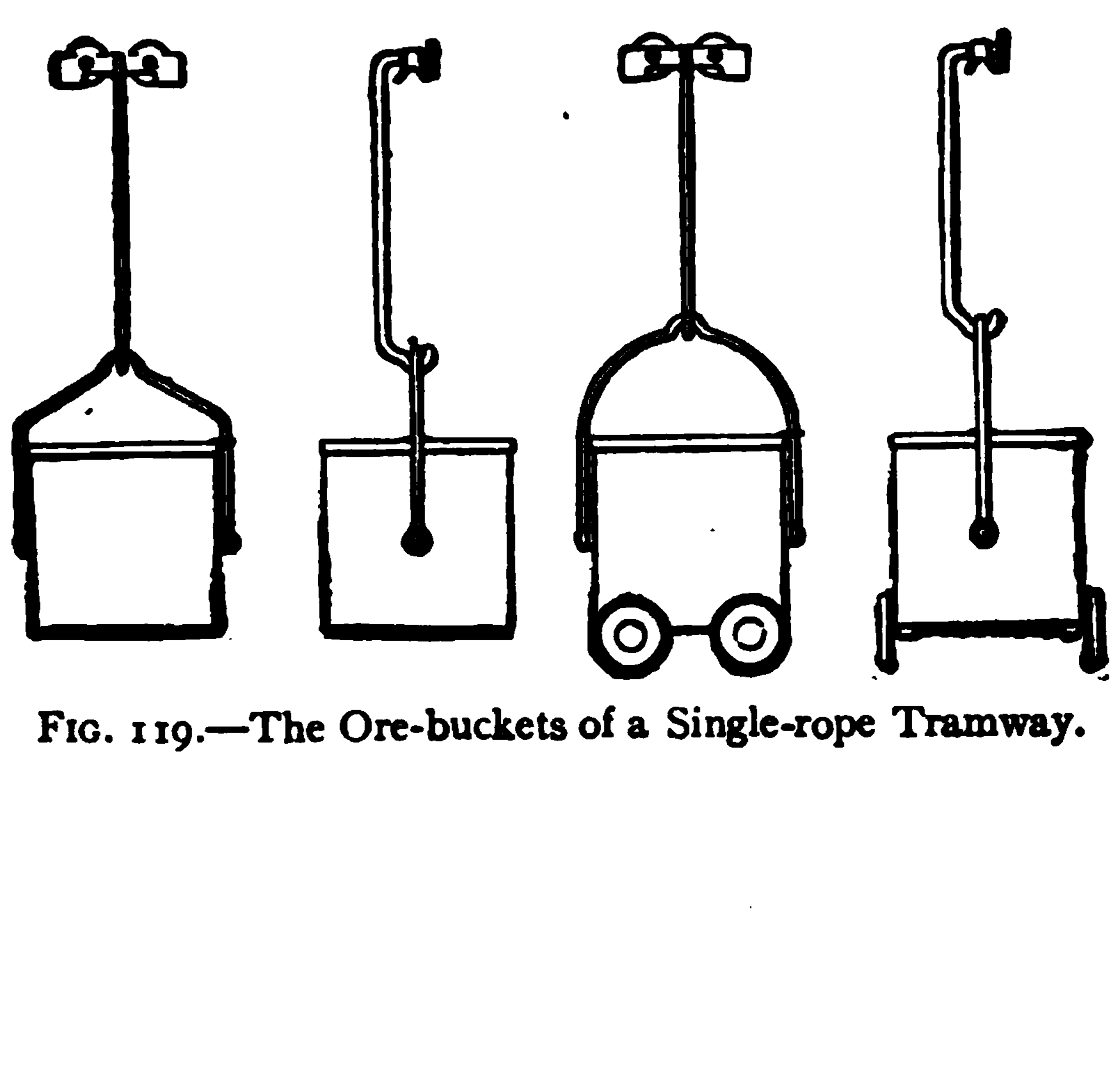

FIG. 119.—The Ore-buckets of a Single-rope Tramway.

to be delivered. Usually they are wrought-iron rectangular buckets holding about 100 lbs. each (Fig. 119). For the transport of very large outputs the buckets may be nearer together than the average 200 feet, or larger, and the rope may be heavier than the ordinary size of ¾ inch. The buckets may be loaded at any point along the line, automatically or by hand. They are unloaded at the lower end by an automatic device in which the carrier strikes a lever, which opens a catch, releasing the swinging bottom and discharging the ore. The bottom is returned to place by a heavy counterpoise weight on the end of a projecting arm and is automatically locked. The hangers are so made that they may pass uninterruptedly over the rims of the supporting sheaves and around the terminal pulleys.

Capacity.—A ropeway running 200 feet per minute, carrying 100 lbs. per bucket every 100 feet, will deliver 60 tons per shift. With a descent sufficient for gravity to supply the power, three men can manage all of its operations. It requires some supervision, and delivers ore at 20 to 35 cents per ton-mile (inclusive of all allowances), and about 60 cents per cord-mile for wood. The line can be completed for $1.30 per foot, and $2000 for the machinery at the terminals. Curves and long stretches increase the first cost; grade does not. The greatest item in the maintenance cost is that for renewals of the clip-hangers. It amounts to $100 a year on a line of half a mile in length.

The Cable Supports.—These are simple frames or towers built up from a rectangular base. The former construction is rigid and not liable to get out of line, nor can its cross-arm be pulled down on the loaded side of the rope. In Figs. 117 and 120 are two typical constructions. The height of the stations depends upon the distance between them. It is about 20 feet when the stations are 200 feet apart. The frames are located at the higher points always when there are any undulations in the grade of the hill, as shorter towers can be used with equal result and the inclination of the rope is more nearly natural. On top of each is bolted a cross-arm, at the ends of which are boxed the carrying-sheaves to support the rope and allow free-

dom of movement. In some instances two sheaves are carried, an upper and a lower one. The object of the upper one is to prevent the rope jumping out of place from its groove in the lower. At a curve the standards are near together and the rope is slightly deflected at each standard to conform to the curve desired. The sharp turns or horizontal angles, as, for example, around bluffs, are also made by using large sheaves. One only is necessary for the turn when the clips holding the buckets project outward from the rope, but two will be required when the clips project inward. In the latter case the ropes cross in passing to the first pulley and also to and from the second pulley.

The end sheaves and carriages are similar to those described for power transmission, with modifications dependent upon local conditions. In Fig. 118 is illustrated the upper frame of the line, while the lower end is illustrated in Fig. 117, on the left.

The Bleichert System.—In the two-rope system there is a pair of separate stationary cables over which numerous trolley carriages, supporting buckets, are drawn by a light endless moving rope, called the traction-rope, to which the buckets are attached by patent grip (Fig. 120). The cables constitute the roadway for the trolleys. The tubs are dumped automatically into a bin or wagon at the lower end of the line.

The several varieties of two-rope systems differ mainly in the mode of suspending the rope. In one system the bucket is attached to a clutch, which seizes the running rope and rails and attaches by friction alone. It locks and releases automatically, and exerts a uniform friction whatever may be the slope of the ropes. The rope permits them to be disconnected and run off on suspended rails when they reach their terminal, or any turnout, on the line. This enables them to be loaded and discharged without interference with the travel.

The double-rope system has a greater capacity than the single rope. It is capable of much heavier traffic, for the individual loads may be as great as 1000 lbs. each, and the distance between the towers 1000 feet, though the average span is 300 feet. The individual loads carried by the Hallidie lines cannot

exceed 150 lbs., nor the distance between the towers 300 feet. The buckets cannot be closer than the distance between the towers. The speed of the rope is limited to 300 feet per minute, because of the danger of the rope jumping off the carriage-sheaves.

TRAMWAY CAR FOR THE TRANSPORTATION OF COAL, ORES, SANDS, &c.

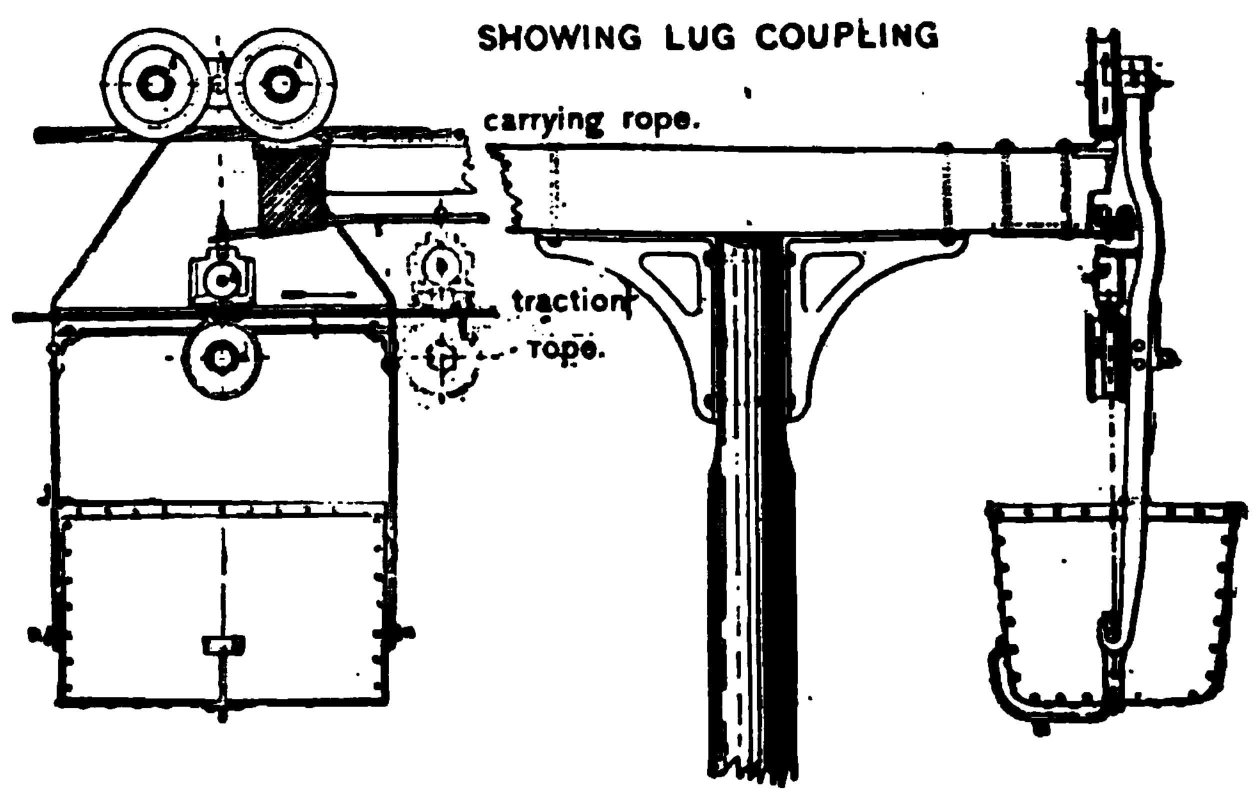

FIG. 120.—A Double-rope Aerial Roadway.

The operating cost of the double-rope system is higher than that of the Hallidie single ropeway, but its capacity is also larger. The materials and supplies, in one instance, for a two-rope system was $1.90 per foot, exclusive of the machinery. The net cost of a single ropeway in the immediate vicinity of the above is $1.21 per foot of line, plus the cost of the two end structures. The cost of repair of either system is not heavy, but the double-rope system is less than that of the single-rope. On one of the former, two miles in length, the cost of maintenance is 1.5 cents per ton out of the total operating cost of 17.5 cents per ton.

A self-acting aerial tramway, like the self-acting underground plane, may, by stretching one or two ropes the full length of the line, serve as a guide for large skips holding a ton or so which are attached to an endless rope. The latter rope is of a length equal to that of the roadway, and is operated from a clip-pulley at the top by gravity with or without power, according as the grade is small or large.

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CHAPTER X.

THE COMPRESSION OF AIR.

Compressed Air is employed in the same manner as steam for motor purposes in driving rock-drills, pumps, locomotives, and coal-cutters. The advantage over steam lies in its transmission without condensation and giving cool, dry, ventilated rooms instead of hot rooms from exhaust where steam is used. It is, however, dearer than the other motor agency. It can be transmitted over great distances where electric transmission is not desirable or economical.

When air is subjected to pressure its volume is proportionately diminished, and the energy thus expended in the compression is retained by the air and capable of being applied as is the ten-

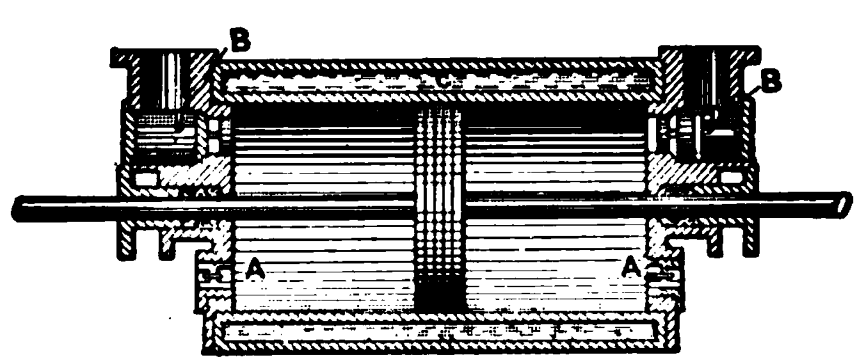

FIG. 121.—A Water-jacketed Air-compressor.

sion of steam. To obtain air at an absolute pressure of 100 lbs. per square inch a unit volume must be reduced to 0.147; at a pressure of 200 lbs. per square inch, to 0.074. In order to obtain the same pressure from steam it must be heated to 338° F. and 388° F. respectively.

The air-compressor is an ordinary cylinder provided with a piston and suitable valves for admitting and delivering air. During

the out-stroke air flows into the cylinder through the inlet-valves *AA*, Fig. 121, which close when the return-stroke commences. The air in the cylinder is then compressed until it opens the delivery-valves, *BB*, through which it is forced into a receiver.

The Weight of Air.—The following table of weights of a cubic foot of air at different temperatures has been calculated on the assumption of a constant pressure of 30 inches of mercury, approximately sea-level pressure:

Temperature *t*.	Weight in Decimals of a Pound.	Temperature *t*.	Weight in Decimals of a Pound.
32	.0809749	120	.0686678
35	.0804831	125	.0680799
40	.0796767	130	.0675020
45	.0788863	135	.0669338
50	.0781113	140	.0663751
55	.0773515	145	.0658256
60	.0766063	150	.0652852
62	.0763122	155	.0647535
65	.0758753	160	.0642305
70	.0751582	165	.0637158
75	.0744544	170	.0632093
80	.0737638	175	.0627108
85	.0730858	180	.0622201
90	.0724202	185	.0617371
95	.0717666	190	.0612614
100	.0711246	195	.0607931
105	.0704941	200	.0603318
110	.0698746	205	.0598775
115	.0692660	212	.0592529

The table on page 309 shows the weight of air at various altitudes above the sea.

Free Air.—Free air is understood as air at the ordinary temperature and an atmospheric pressure of 14.7 lbs. per square inch or 2116.8 lbs. absolute per square foot. By "absolute pressure" is meant the pressure, above the vacuum as distinguished from gauge pressure, which is measured above the atmosphere by the gauge. Absolute temperature, at Fahrenheit scale, is the reading of the thermometer, in Fahrenheit, plus 461°, and is represented by τ. At $-461°$ F. there is no pressure. The absolute temperature is 0°.

TABLE OF ABSOLUTE PRESSURES, BOILING-POINTS, ETC., AT DIFFERENT HEIGHTS ABOVE SEA-LEVEL.

1	2	3	4	5	6	7
Height above Sea-level, Feet.	Barometer, Inches of Mercury.	Boiling-point, Degrees Fahr.	Absolute Pressure, Pounds.	Weight of 1 Cubic Foot of Air at 60°, Pounds.	Volume of Air Equal to 1 Cubic Foot of Free Air at Sea-level.	Volume of Free Air at Sea-level Equal to 1 Cubic Foot at Given Altitude.
0	30	212	14.7	.0765	1	1
512	29.42	211	14.41	.07499	1.02	.98039
1,025	28.85	210	14.136	.07356	1.04	.96154
1,539	28.29	209	13.86	.07213	1.06	.9434
2,063	27.73	208	13.587	.07071	1.08	.9259
2,589	27.18	207	13.318	.0693	1.10	.90909
3,115	26.64	206	13.054	.06793	1.12	.89285
3,642	26.11	205	12.794	.06658	1.14	.87719
4,169	25.59	204	12.539	.06525	1.17	.8547
4,697	25.08	203	12.289	.06395	1.19	.8403
5,225	24.58	202	12.044	.06267	1.22	.8197
5,764	24.08	201	11.799	.0614	1.24	.8064
6,304	23.59	200	11.559	.06015	1.27	.7874
6,843	23.11	199	11.324	.05893	1.29	.7752
7,381	22.64	198	11.094	.05773	1.32	.75757
7,932	22.17	197	10.863	.0565	1.35	.74074
8,481	21.71	196	10.638	.05536	1.38	.7246
9,031	21.26	195	10.417	.05421	1.41	.7092
9,579	20.82	194	10.202	.05309	1.44	.6944
10,127	20.39	193	9.99	.05199	1.47	.6802
10,685	19.96	192	9.78	.0509	1.50	.6666
11,243	19.54	191	9.57	.0498	1.53	.6536
11,799	19.13	190	9.37	.0488	1.56	.64102
12,367	18.72	189	9.17	.0477	1.60	.625
12,934	18.32	188	8.98	.0467	1.63	.6135
13,498	17.93	187	8.78	.0457	1.67	.6
14,075	17.54	186	8.59	.0447	1.71	.5848
14,649	17.16	185	8.41	.0437	1.74	.5747

The volume of air may be altered by a change of temperature or of pressure. It increases with a rise in temperature or a reduction of pressure. Correspondingly it decreases for a diminished temperature or an increased pressure.

Adiabatic Compression.—When compressed or expanded by the application of force, air suffers a change of temperature. If the walls confining the air be absolute non-conductors of heat, the increase in temperature and of pressure may be definitely known.

The relation between the pressure, volume, and temperature is expressed by the following equations, in which P = the initial pressure of the air in pounds per square foot; V = the volume at that pressure in cubic feet; and τ = its absolute temperature P, V, and τ correspondingly represent the pressure, volume, and temperature at the end of the operation.

$$\frac{\tau_1}{\tau} = \left(\frac{V}{V_1}\right)^{0.408} = \left(\frac{P_1}{P}\right)^{0.29},$$

and

$$PV^{1.408} = P_1 V_1^{1.408}.$$

$$\log \tau_1 = \log \tau + 0.29 \log P_1 - 0.29 \log P;$$
$$\log \tau_1 = \log \tau + 0.408 \log V - 0.408 \log V_1;$$
$$\log P + 1.408 \log V = \log P_1 + 1.408 \log V_1.$$

Thus, if a pound of free air at 60° be compressed to 90 lbs. absolute, it will attain a final temperature of 881° absolute or 420° F., the final volume becoming 0.276 of the original. This is known as adiabatic compression.

$$\log \tau_1 = 2.716838 + 0.29 \times 1.954243 - 0.29 \times 1.176091,$$

whence $\tau_1 = 881$.

Isothermal Compression.—If the air be compressed within walls which are perfect conductors and the compression be a slow one, the heat generated will be absorbed or radiated as fast as developed and the air, originally at 60°, will remain at 60°, the process being isothermal. At the final pressure of 90 lbs. absolute the volume would be only 0.1640 of the original. Thus isothermal compression furnishes a smaller volume at a lower temperature than is attained in adiabatic compression.

The relation of volume, temperature, and pressure is represented by the equation $PV = 53.18\tau$. Thus the volume of air varies inversely as the pressure when the temperature is constant; the absolute pressure varies directly with an absolute temperature if the volume remains constant; and the volume varies as the absolute temperature so long as the pressure remains constant.

TABLE OF VOLUMES, MEAN PRESSURES, TEMPERATURES, ETC., IN THE OPERATION OF AIR-COMPRESSION FROM 1 ATMOSPHERE AND 60° FAHR.*

1	2	3	4			7	8	9	10	11
Gauge Pressure.	Absolute Pressure.	Pressure in Atmospheres.	Volume with Air at Constant Temperature.			Mean Pressure per Stroke, Air Not Cooled.	Mean Pressure During [illegible] Pressure.	Mean Pressure During Compression Only, Air Not Cooled.	Final Temperature, Air Not Cooled.	Gauge Pressure.
0	14.7	1	1	1	0	0	0	0	60	
1	15.7	1.068	.9363	.95	.96	.975	.43	.44	71	1
2	16.7	1 136	.8803	.91	1.87	1.91	.96	.96	80.4	2
3	17 7	1.204	.8305	.876	2.72	2.8	1.4	1.41	88.9	3
4	18.7	1.272	7861	.84	3.53	3.67	1.84	1.86	98	4
5	19.7	1.34	7462	81	4.3	4.5	2.22	2.26	106	[illegible]
10	24.7	1.68	5952	.69	7.62	[illegible].27	4 14	4.26	145	10
15	29.7	2.02	495	.606	10.33	11.51	5.77	5.99	178	15
20	34 7	2.36	.4237	.543	12.62	14.4	7.2	7 58	207	20
25	39.7	2.[illegible]	.3703	.494	14 59	17.01	8 49	9 05	234	25
30	44.7	3.04	.3289	.4638	16.34	19.4	9.66	10.39	255	30
35	49 7	3.381	.2957	.42	17 92	21.6	10 72	11 59	281	35
40	54.7	3 721	.2687	.393	19 32	23.66	11.7	12.8	302	40
45	59.7	4 061	.2462	.37	20 52	25.59	12.62	13.95	321	45
50	64.7	4 401	.2272	35	21 79	27.39	13 48	15 05	339	50
55	69.7	4 749	2109	.331	22 77	29.11	14 3	15 98	357	55
60	74.7	5 081	.1968	.3144	23.84	30.75	15.05	16 89	375	60
65	79.7	5.423	.1844	.301	24 77	31 69	15 76	17 88	389	65
70	84.7	5.762	1735	.288	26	33.73	16.43	18.74	405	70
75	89.7	6.102	.1639	.276	26.65	35 23	17.09	19.54	420	75
80	94.7	6.442	.1552	.267	27.33	36 6	17.7	20 5	432	80
85	99.7	6 782	.1474	.2566	[illegible].05	37.94	18.3	21 22	447	85
90	104 7	7.122	.1404	.248	28.78	39.18	18.87	22	459	90
95	109.7	7 462	.134	.24	29 53	40.4	19.4	22.77	472	95
100	114.7	7.802	.1281	.232	30 07	41.6	19 92	23 43	485	100
105	119.7	8.142	.1228	2254	30.81	42.78	20.43	24.17	496	105
110	124.7	8.483	.1178	2189	31.39	43 91	20 9	24.85	507	110
115	129.7	8 823	1133	.2129	31 98	44.98	21 39	25.54	518	115
120	134.7	9.163	.1091	.2073	32.54	46.04	21.84	26 2	529	120
125	139.7	9.503	.1052	.202	33.07	47.06	22.26	26.81	540	125
130	144.7	9 843	.1015	1969	33 57	48.1	22 69	27 42	550	130
135	149.7	10.183	0981	.1922	34 05	49.1	23 08	28.05	560	135
140	154.7	10.523	095	.1878	34.57	50.02	23 41	28 66	570	140
145	159.7	10.864	.0921	.1837	35 09	51	23 97	29 26	580	145
150	164.7	11.204	.0892	.1796	35 48	51.89	24 28	29.82	589	150
160	174.7	11 88	0841	.1722	36.[illegible]	53.65	24 97	30.91	607	160
170	184.7	12.56	.0796	1657	37 2	55 39	25.71	32 03	624	170
180	194.7	13 24	0755	.1595	37.96	57 01	26.36	33.04	640	180
190	204.7	13 92	.0718	.154	38.68	58 57	27.02	34 06	657	190
200	214.7	14 6	0685	.149	39 42	60 14	27.71	35.02	672	200

*From Richards' "Compressed Air," page 21.

The table on page 311 shows the relation between volume, pressure, and temperature in adiabatic and isothermal compression.

The columns 4, 8, and 6 respectively express the relations assumed by one unit of air when compressed without change of temperature to the stated pressures. Column 8 shows the mean effective resistance during the period of compression which is offered by the air to the point of delivery only. Column 5 contains the ratio of volumes of air compressed adiabatically to given pressures; column 9, the mean effective resistance during the compression period; and column 6, the mean effective resistance during the entire stroke, including the period of delivery.

EXAMPLES.—1. Let it be desired to determine the final volume of 1 lb. of free air at a temperature of 62° F. compressed to 80 lbs. gauge. Then

$V=13.08$; P is $14.7\times 144=2116.8$; $P_1=94.7$; and $\tau=62+461=523°$.

Isothermally,

$$2116.8\times 13.08=27,729=P_1V_1. \quad P_1V_1=94.7\times 144\times V_1.$$

Whence

$$V=2.027 \text{ cu. ft.}$$

Adiabatically,

$$2116.8\times(13.08)^{1.408}=63,110=P_1(V_1)^{1.408}; \quad 94.7\times 144\times(V_1)^{1.408}=63,110.$$

Whence

$$V_1=3.492.$$

According to the preceding table the volumes are respectively (columns 4 and 5) $0.1552\times 13.08=2.027$ cu. ft. and $0.267\times 13.08=3.49$ cu. ft.

What is the final temperature in the latter case?

$$\log \tau=\log 523+0.29\log 94.7-0.29\log 14.7=2.953151.$$
$$\tau=897° \text{ abs.} =436° \text{ F.}$$

According to the table, column 10, τ would have been 432° F. from 60° F.

2. Required the temperature of a pound of air which at 50 lbs. abs pressure occupies a volume of 10 cu. ft.

$$PV=50\times 144\times 10=53.18\tau;$$

whence

$$\tau=1353° \text{ absolute}=892° \text{ F.}$$

Air Indicator Cards. — In Fig. 122 are indicator cards steam- and air-cylinders; they show also the adiabatic and isothermal compression. On the horizontal the volumes are measured from the right, and on the vertical lines the corresponding pressures. It is noticeable that the adiabatic curve rises more rapidly than the isothermal; in other words, its pressure increases more rapidly. Again, the volumes in the former case are correspondingly larger for a given pressure, as may be seen by comparing volumes obtained by cutting the two curves by some horizontal line. This may also be noted by comparing the volumes in columns 5 and 4 of table page 311.

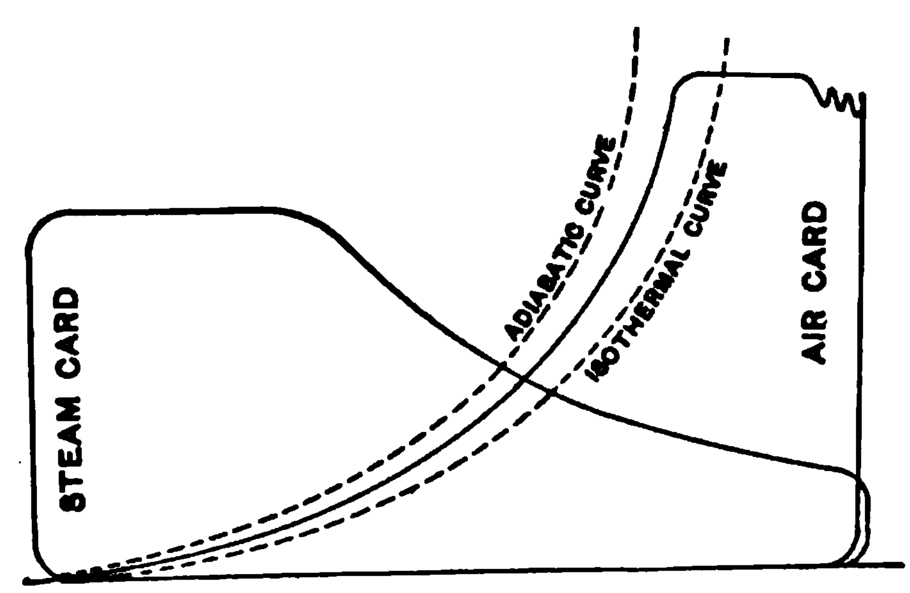

FIG. 122.—Indicator Cards from Steam- and Air-cylinders.

The initial pressure of the steam in Fig. 122 is 58 lbs. gauge, and the cut-off 0.3. The final pressure of the delivery air is 80 lbs. gauge. The mean effective pressure for the entire card, including the period of d:scha ge represented by the top horizontal line, may be obtained in columns 7 and 6 respectively. These mean effective pressures represent the resistance to compression offered by the air under the conditions assumed. Thus, for example, according to the table the mean effective pressure of the entire air card, assuming isothermal compression to be 94.7 lbs. pressure absolute, 80 lbs. gauge, would be 27.33 lbs. per square inch; if carried to the same degree adiabatically, 36.6 lbs. per square inch. During the compression of the strokes only, the mean effective pressures are respectively 20.5 and 17.7 lbs. per square inch.

The Work of Adiabatic Compression of w pounds of air is

$$W_a = 183.45\,(\tau_1 - \tau)w = 3.45w(P_1V_1 - PV).$$

The Work of Isothermal Compression of w pounds of air is

$$W_i = P(V+v)\ \text{hyp log}\ \frac{V+v}{V_1+v}w - (P_1 - P)vw,$$

in which v is the volume of clearance in the cylinder.

TABLE OF HYPERBOLIC LOGARITHMS.

(Base 2.72.)

Hyp log = 2.3026 × common logarithm.

Number.	Logarithm.	Number.	Logarithm.	Number.	Logarithm
1.01	.009	2.10	.741	4.60	1.526
1.02	.019	2.20	.788	4.70	1.547
1.03	.029	2.30	.832	4.80	1.568
1.04	.039	2.40	.875	4.90	1.589
1.05	.048	2.50	.916	5.00	1.609
1.06	.058	2.60	.955	5.10	1.629
1.07	.067	2.70	.993	5.20	1.648
1.08	.076	2.80	1.029	5.30	1.667
1.09	.086	2.90	1.064	5.40	1.686
1.10	.095	3.00	1.098	5.50	1.704
1.11	.104	3.10	1.131	5.60	1.722
1.12	.113	3.20	1.163	5.70	1.740
1.13	.112	3.30	1.193	5.80	1.757
1.14	.131	3.40	1.223	5.90	1.774
1.15	.139	3.50	1.252	6.00	1.791
1.20	.182	3.60	1.280	6.20	1.824
1.25	.223	3.70	1.308	6.40	1.856
1.30	.262	3.80	1.335	6.60	1.887
1.40	.336	3.90	1.360	6.80	1.916
1.50	.405	4.00	1.386	7.00	1.945
1.60	.470	4.10	1.410	7.20	1.974
1.70	.530	4.20	1.435	7.40	2.001
1.80	.587	4.30	1.458	7.60	2.028
1.90	.641	4.40	1.481	7.80	2.054
2.00	.693	4.50	1.504	8.00	2.079

The following table will facilitate calculation of engine-power required for compression. These values are to be divided by m, the modulus of the compressor, which is for 0.50 to 0.70, to

determine the steam-power required to drive the piston against friction.

TABLE SHOWING THE HORSE-POWER REQUIRED TO COMPRESS AND DELIVER ONE CUBIC FOOT OF FREE AIR PER MINUTE TO VARIOUS GAUGE PRESSURES; ALSO THE POWER REQUIRED TO COMPRESS AND DELIVER ONE CUBIC FOOT OF AIR AT THE GIVEN PRESSURE.

1 Gauge Pressure.	Compressing One Cubic Foot of Free Air per Minute to Given Pressure.		Delivering One Cubic Foot per Minute of Air Compressed to the Pressure Given.	
	2 Compression at Constant Temperature	3 Compression without Cooling.	4 Compression at Constant Temperature.	5 Compression without Cooling.
5	.01876	.01963	.02514	.0263
10	.03325	.03609	.05586	.06399
15	.04507	.05022	.09105	.10145
20	.05506	.06283	.12994	.14829
25	.06366	.07422	.17191	.20043
30	.0713	.08464	.21678	.25734
35	.0782	.09425	.26445	.31872
40	.084305	.10324	.31375	.38422
45	.08954	.11166	.36368	.45353
50	.09508	.11952	.41848	.52605
55	.09936	.12702	.47112	.60227
60	.10402	.13418	.52855	.68181
65	.10808	.14028	.58612	.76079
70	.11245	.14718	.64812	.8483
75	.11629	.15373	.70952	.93795
80	.11926	.15971	.76843	1.02906
85	.1224	.16555	.83039	1.1231
90	.12558	.17096	.89444	1.2176
95	.12886	.17629	.96164	1.3148
100	.13121	.18153	1.0243	1.4171

EXAMPLES.—1. Required the number of units of work necessary to compress 80 lbs. of free air from 14.7 lbs. absolute pressure and 60° F. to a pressure of 88.2 lbs. per square inch. Here $P=2116.8$; $V=80\times 1308=1046$; $V_1=0.166\times 1046=174.3$.

Neglecting friction and the value for the clearance in the cylinder, $W_t=2116.8\times 1046\times 1.79=3{,}963{,}320$ ft.-lbs. Compressed adiabatically, $\tau=60+461=521°$, and τ_1, according to the previous table, is nearly 881°, or by computation $\log \tau_1=\log 521+0.29 \log (88.2\times 144)-0.29 \log (2116.8)=2.94254$. Whence $\tau_1=876.1°$, $W_a=183.5(\tau_1-\tau)\times 80=5{,}211{,}500$ ft.-lbs. By the preceding table the work is respectively 121.65 and 160.77 h.p. for a final compression to 89.7 lbs. absolute.

2. 100 cubic feet of air per minute are to be compressed to 59 lbs. gauge

without cooling. Required the work expended in the operation. The initial temperature of the free air is 60°; then $P=14.7$; $P_1=64.7$; $V=100$; $V_1=35.00$; $\tau=521°$; $\tau_1=339°$ F.

Neglecting clearance, the value for the work per cubic foot, according to the table, is 0.11952 h.p., and for 100 cubic feet 11.952 h.p.

$$W_a=183.45(800-521)\frac{100}{13.08}=11.9\times33,000 \text{ ft.-lbs.}$$

If the compression were conducted with perfect cooling, then, according to the table, there would be required 9.5 horse-power and the final volume V_1 would be 22.7.

$$W_i=2116.8\times100 \text{ hyp log } 4.401=9.504\times33,000 \text{ ft.-lbs.}$$

3. What should be the size of the cylinders to compress 100 lbs. free air per minute from 60° F. to 100 lbs. gauge pressure? $\tau=521°$ abs.; $P_1=114.7\times144$; $V=1308$ cubic feet; $v=7$ per cent V; $w=100$; $\tau_1=946°$ F.

$$W_a=183.45(946-521)100=7,797,000 \text{ ft.-lbs.};$$

$$W_i=2116.8(1.07\times1308) \text{ hyp log } \frac{13.99}{1.765}-(14,400)91.5=4,843,300 \text{ ft.-lbs.}$$

If the modulus, m, be taken at 0.70, the work to be supplied the steam-piston is 11,138,571 ft.-lbs. and 6,919,000 ft.-lbs. respectively. With an actual capacity of cylinder 0.90 that of the apparent capacity, there being 80 strokes per minute, the cylinder capacity is, for isothermal compression, 16.35 cubic feet. With 7 per cent clearance the cushioned air will correspond to $0.302V$ of free air. Each piston displacement is then $(1.302-0.07)8.175=$ 10.07 cubic feet. The cylinders are $23\frac{3}{4}''\times36''$ stroke each.

The mean effective resistance is 30.07 lbs. per square inch. The steam-cylinder must be $12\frac{1}{2}''\times36''$ if the average cut-off is $\frac{1}{2}$, initial pressure 120 lbs. gauge, and the back pressure 5 lbs. Then, Chapter V, 3 per cent clearance,

$$\text{m.e.p.}=0.8658(120+14.7)-5=116.6;$$

$$H=\frac{6,919,000}{33,000}=209.7;$$

whence

$$k=12.54 \text{ inches} \quad \text{and} \quad s=3 \text{ feet.}$$

The Mean Resistance to Compression of dry air during compression can be ascertained. Let P_1 be the final pressure and P the initial pressure; then

$$\text{m.e.p.}=3.45\,P\left[\left(\frac{P_1}{P}\right)^{0.29}-1\right].$$

Fig. 123 is a diagram showing on the right the temperatures attained by air when compressed adiabatically to the degree indicated at the bottom of the figure, and on the left the work in foot-pounds expended during compression. The difference in power consumption by adiabatic compression and isothermal compression is pictorially revealed. Thus a point on the isothermal compression line corresponding to 6 atmospheres indicates the work expended on one pound as 50,000 ft.-lbs.; upon

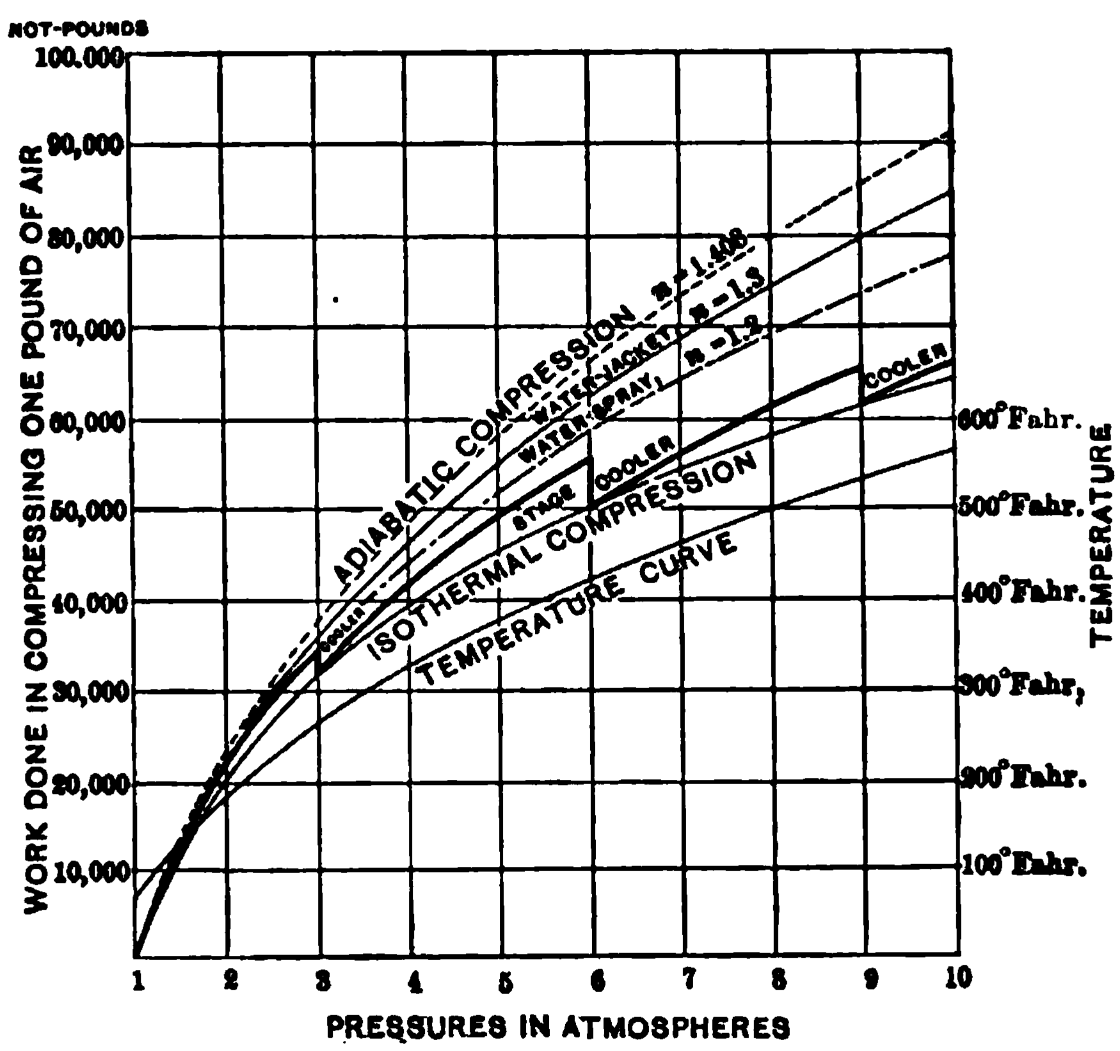

FIG. 123.—Diagram Showing the Work Done During Air-compression under Various Conditions.

one pound at the same pressure adiabatic there has been expended 73,300 ft.-lbs. and the temperature attained is 460° F.

Cooling the Air During Compression.—Isothermal compression is more desirable to attain than adiabatic compression, not only because the power required is less, but also because the high temperature of the latter is particularly injurious to the machinery. It is an obstacle to rapid running, as proper lubrication cannot be maintained.

Isothermal compression is too slow. Moreover, the cylinder walls are never perfect conductors. So the air is taken as cool

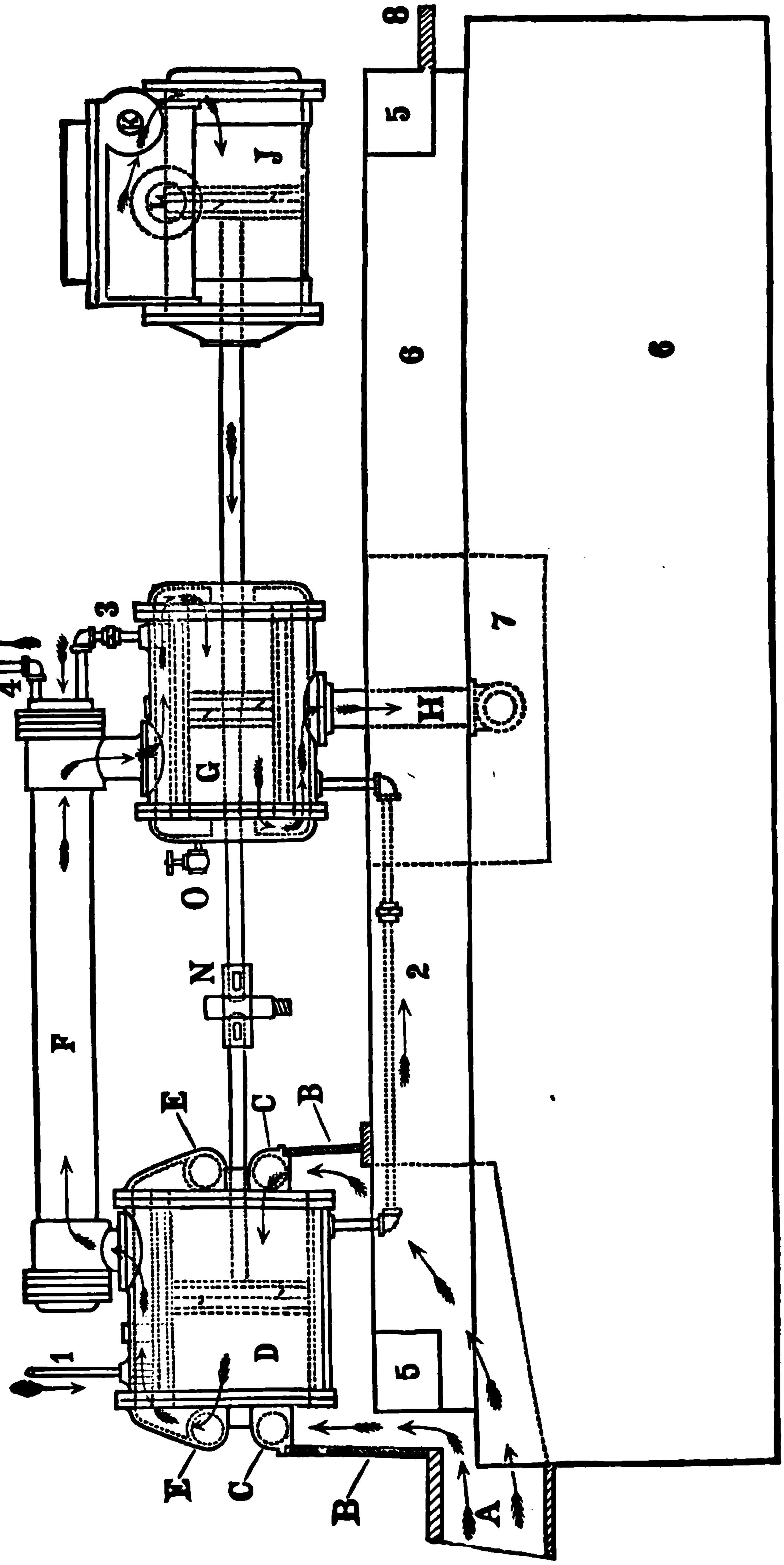

FIG. 124.—Two-stage Air-compressor with Intercooler, *F*. Course of the jacket-cooling water 1, 2, 3, 4.

as possible—chilled if may be—and kept cool during all stages of the compression by artificial means. As the air will cool rapidly after compression, producing a great loss by radiation from cylinders, receivers, conducting pipes, and all intermediate appliances before reaching the motor, and as it is impossible to retain this heat in spite of every precaution, it is desirable to extract the excessive heat as promptly as possible. So the supply of air is taken from outdoors at the coolest side of the building, effecting a saving of at least 2 per cent in power by this choice of site, which may be 10° cooler than on the other side. It may be noted, too, in the table and the diagram, Fig. 123, that the increment of heat is greater in the earlier portion of the stroke. During the first 15 lbs. increase the temperature rises 118° F., but during the increment from 90 to 105 lbs. the rise is only 37° F.

Water-cooling.—Cooling is effected by a circulating current of water in order to maintain an isothermal compression, in which case there will be extracted from the air under compression an amount of heat corresponding to the area of the space between the two curves in Fig. 123, or an amount of work represented by the difference in the vertical lines between the isothermal and adiabatic curves in diagram Fig. 122. The cooling is a direct waste, and heat once removed from the air is never returned to it. The absorption of heat is accomplished by cold air circulating through a jacket, *j*, Fig. 128, surrounding the cylinder. In Fig. 124, the course of the water through it is represented by the arrows and passages numbered 1, 2, 3, and 4.

A fine spray of water injected into the cylinder would be a more effective method of cooling, were it not so objectionable because of its corrosive action on the cylinder walls and its freezing when, at some later stage, it is used expansively in the motor. The spray-injectors are more efficient coolers than water-jackets, as shown in the diagram Fig. 125, the former curves being lower than the latter.

The circulating water should be the coolest attainable, and the cylinder lining as perfect a conductor as possible, to extract the appreciable amount of heat.

Two-stage Compression.—The very short time of contact between the water and the hot air may be increased by dividing the air-compression between two cylinders (Fig. 126), and in addition passing it through the intercooler. This latter is very efficient, saving as much as 10 per cent of the power, which almost counterbalances the friction of the machine. This compound, or two-stage, compression gives nearly treble the time of contact for the circulating waters. In the larger cylinder the air is compressed to one third or one fourth its volume, and in the

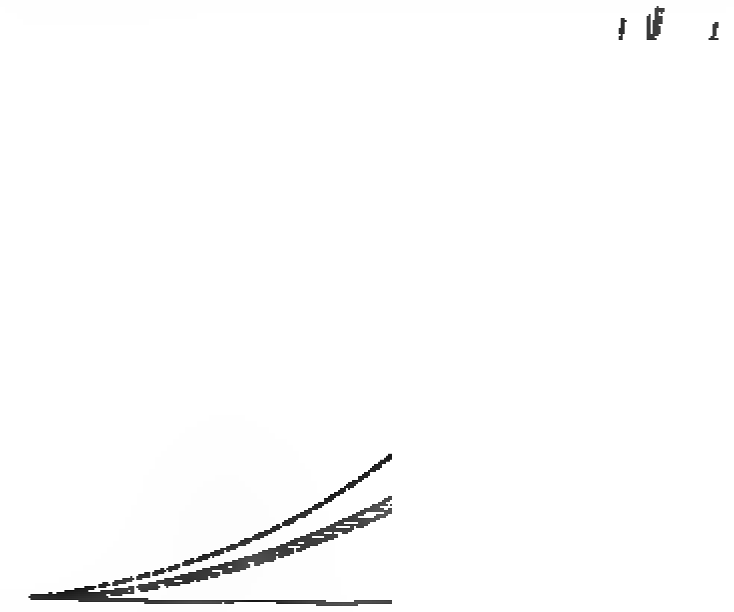

FIG. 125.—Combined Indicator Diagrams from a Two-stage Air-compressor.

small cylinder to that required. The intercooler is essentially a cylinder with small brass pipes through which water circulates and around which air flows from the low- to the high-pressure cylinder. In this way a stepped curve, diagram Fig. 125, is obtained indicating economical results approaching an isothermal compression. From 1½ to 3 horse-power are saved over the single-stage compression for each 100 cubic feet of free air at the ordinary degrees of compression used in mining. Two-stage compression is not advisable below 4 atmospheres final pressure, but is imperative for high degrees of compression. But the greater the degree of compression, the greater is the amount of abstracted heat and the more serious becomes the loss of power. For 1, 2, 3, 4, 5, and 6 atmospheres, gauge reading, the losses are 28%, 37%, 46%, 50%, 53%, and 56%, respectively,

of the original power. Economic work is best obtained by operating at as low pressure as is consistent with the work.

HORSE-POWER NECESSARY TO COMPRESS 100 CUBIC FEET OF FREE AIR TO VARIOUS PRESSURES AND WITH TWO-, THREE-, AND FOUR-STAGE COMPRESSORS.

Gauge Pressure.	Horse-power Necessary.			Gauge Pressure.	Horse-power Necessary.		
	Two-stage.	Three-stage.	Four-stage.		Two-stage.	Three-stage.	Four-stage.
100	15.7	15.2	14.2	900	36.3	33.7	31.0
200	21.2	20.3	18.8	1000	37.8	34.9	31.8
300	24.5	23.1	21.8	1200	39.7	36.5	33.4
400	27.7	25.9	24.0	1400	41.3	37.9	34.5
500	29.4	27.7	25.9	1600	43.0	39.4	35.6
600	31.6	29.5	27.4	1800	44.3	40.5	36.7
700	33.4	31.2	28.9	2000	45.4	41.6	37.8
800	34.9	32.5	30.1	2500	....	43.0	39.0

The Work of Compressing Moist Air is less than that required during the compression of dry air, nor do the mean net resistance and the temperature increase as rapidly.

RISE IN TEMPERATURE AND WORK OF COMPRESSION OF DRY AND MOIST AIR.

Absolute Pressure.	Temperature.		Work on One Pound.	
	Dry.	Moist.	Dry.	Moist.
14.7	68° F.	68° F.	——	——
22	133.8	94	13,300	13,200
29.4	185.9	111	23,500	22,500
36.7	229.5	124	30,500	29,000
44.1	266.7	135	37,000	35,000
51.4	300.2	145	43,200	40,600
58.8	330.1	153	48,500	45,000
73.5	383.5	167	58,500	52,500
88.2	428.9	179	67,160	60,000

The mean effective resistance or pressure during compression or expansion of moist air is

$$\text{m.e.p.} = 88.2\left\{\left(\frac{P_1}{P}\right)^{0.166} - 1\right\},$$

and

$$\frac{\tau_1}{\tau} = \left(\frac{P_1}{P}\right)^{0.166}.$$

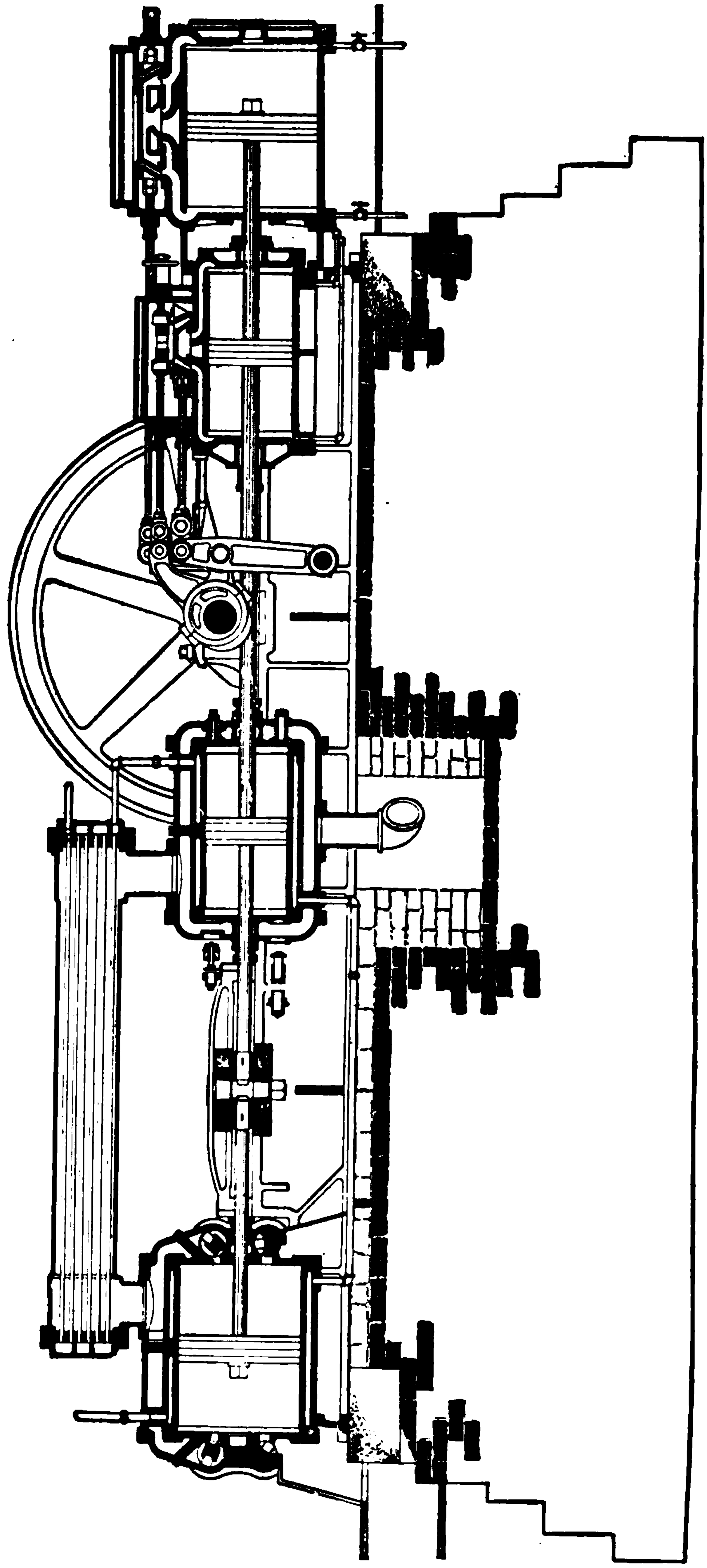

FIG. 126.—Section of a Two-stage Compressor.

The Air-compressor.—The simple principle of air-compression is difficult of execution. To obtain a compact high-speed, uniform, rapid-cooling, efficient compressing engine is not easy.

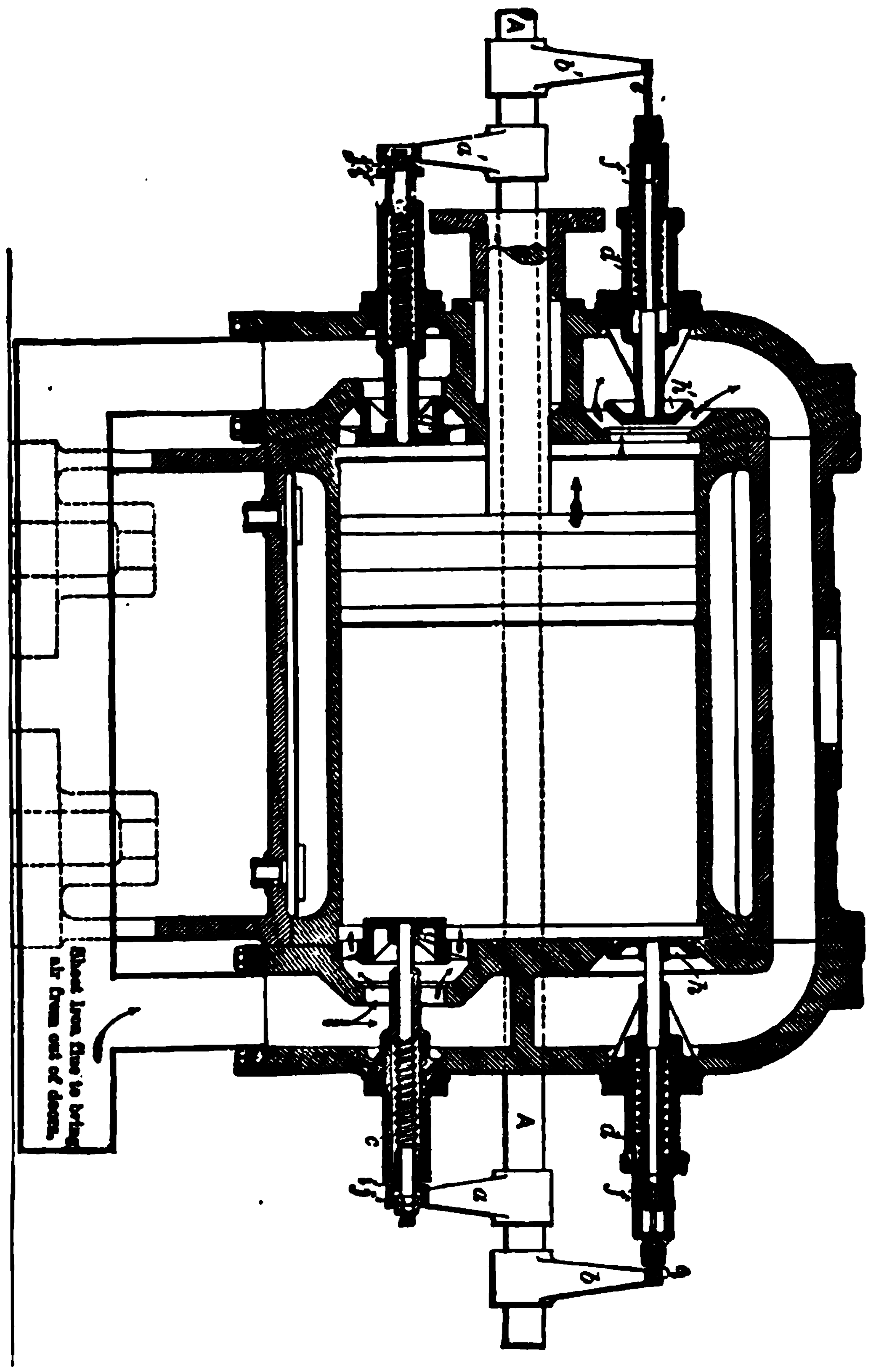

FIG. 127.—Rand Air-cylinder Showing Poppet-valves.

The essentials are partly secured in various ways by the several successful patterns now on the market.

The piston of the air-compressor may be driven directly by a steam-cylinder at the other end of the same rod (in tandem),

or "straight line," or it may be alongside of and joined by a common cross-head with the steam-piston. Such an engine is called a "cross" air-compressor. It may be driven by a belt or by a reducing-gear from a shaft actuated by electricity or steam-power, or an impulse-wheel mounted directly upon the main shaft of the compressor. The cylinders are usually horizontal and

Fig. 128.—A Water-jacketed Air-compressor, Ingersoll Pattern.

may be single or duplex, and the air may be compressed in one stage with one cylinder, or in two stages, corresponding to simple and compound steam-engines. The duplex engines are simple, inhaling and compressing separate volumes of air. They have none of the merits of the compound or two-stage engines, in which the large low-pressure cylinder expels its air into the small high-pressure cylinder. The cranks of a duplex compressor are at quarters to equalize the rotary effort.

The frame, compared with that of the steam-engine, is very solid and on heavy base-plate Compressors may be had sectional for convenience in transportation. They are fitted with automatic throttling governors to alter the steam pressure according to the demand for power, and are so connected that when the delivery air attains an excessive pressure it will operate on the piston, which raises the regulator and closes the governor-valve, thus reducing compressor speed. Their action is very much like that of the automatic damper regulator mentioned in Chapter IV.

FIG. 129.—Two-stage Compressor with Rotary Valves.

High speed to the piston is advantageous for the economy of steam and for capacity, but because of the rapid wear and the difficulties with large valve areas, as well as the inordinate resistance developed thereby, the velocity is less than 300 feet per minute, except in the larger sizes.

The Burleigh is upright, its air-cylinder is single-acting, and its peculiarity lies in the admission of steam one eighth of a stroke before the air. The Waring has a bonnet, or conical valve, like that in Fig. 127, whose pistons are moved by a rocker on the fly-wheel shaft, the steam-cylinder being set at an angle to the horizontal air-cylinder. The Clayton has the usual poppet-valve, and is a compact machine, with its fly-wheel centrally located. The Delamater has an important contrivance for dropping the discharge-valve from its seat. This form is very heavy. The Sullivan, Ingersoll, Norwalk, and Rand are the popular pneumatic machines.

The Valves of the Compressor are of the poppet, spindle, or ring pattern. Whatever their form, they should open quickly, have a full lift, and be ample in size. Large inlet-valves offer little difficulty, though for a short time they are subject to full reservoir pressure. An unrestricted entry for the air is obtained easily by the use of poppets held by springs. The Ingersoll-Sergeant compressor admits air through a hollow piston and rod (Fig. 128). This provides a very liberal inlet area, *G*, enables the cylinder-covers, *J*, to be completely water-jacketed, and leaves more room for the discharge-valves, *H*. The Norwalk employs a rotary valve. The inlet-valves of the Rand are shown at *g*, *g'*, Fig. 127. They are provided with guards that prevent their falling into the cylinder.

The valves should be positive, and this the poppet attains, though the tendency to "chattering" is the serious objection to it, particularly for discharge-valves. This arises from the two opposing effort—one, of the air, to open, and the other, of the spring, to close the valve. The valve-gear shown in Fig. 124 does away with this trouble in the high-pressure engines; the arms, *a*, *b*, relax the spring pressure and allow of the valve rising

full-lift without dancing. Poppet-valves can hardly be improved upon for low pressures, though their springs in time lose elasticity and open too soon. This reduces their efficiency, as also does any slip of the valves. In the Norwalk pattern (Fig. 129) a positive discharge is obtained by moving the valve by cams, such that it remains at rest till the pressure is sufficient to open it quickly. A difficulty about this, it would seem, is that, as the reservoir pressure constantly varies (unless perfectly regulated), the valves must receive constant attention. This may be corrected by an automatic governor like the Corliss release, which will open the valves at different points in the stroke as desired.

The discharge-valves require careful construction, for their leakage is equal to a large clearance space. They are made large, and, to prevent inordinate frictional loss and wear, are as numerous as possible. An excess of air pressure over the receiver pressure is necessary to open the valves and expel the air. This unavoidable loss has an important bearing upon the uniformity of speed.

The Effect of the Clearance Space.—The large clearance space between the piston and the cylinder-head at the end of the stroke constitutes one inevitable source of reduction of cylinder capacity. Not only is the compressed air filling that space never discharged, but on the forward-stroke it will expand and fill a volume that should have been occupied by fresh atmospheric air which is being inhaled.

If the air in the clearance is compressed on the return-stroke to 60 lbs. and occupies 7 per cent of the cylinder volume, it is evident that in the forward-stroke it expands to five times the original volume, or nearly one-third that of the cylinder, before any fresh cool air whatever can be admitted. The effective volume of the cylinder is therefore materially reduced. Moreover, this air is hot, and, the cylinder also being warm, such inlet air as does enter is expanded to correspondingly reduce the capacity of the cylinder. The clearance space cannot be made smaller than $\frac{1}{16}$ of an inch, and is usually much more. The only remedy for this

loss is an increased length of stroke or compounding the cylinders. At 75 lbs. absolute pressure a single cylinder must be three times as long as a compound having the same clearance loss.

The friction of the air-piston taken with that of the valves is about equal to 10 per cent of the work of the engine, and may reach 25 per cent when including the losses in the steam-engine.

The Horse-power of the Compressor.—This is determined, in the same general manner as for the steam-engine, by the indicator cards or by assuming a mean effective pressure, obtained from column 6 of the table of volumes, and substituting the same in the general expression for horse-power. The mechanical efficiency of the air-compressor m must be taken at not more than 60 per cent for the one-stage and 70 per cent for the two-stage cylinder, measured in the terms of the energy possessed by the compressed air delivered from the machine, compared with the steam-motor power.

By inspection of the diagram Fig. 122 it is seen that the air resistance is atmospheric at the beginning of the stroke when the steam-power is at the maximum. Toward the end of the stroke it is at a maximum, while the expanded steam is at its minimum pressure. This large engine excess at the beginning, and resistance at the end, necessitate the use of heavy fly-wheels. Indeed, the fluctuating stresses to which the entire compressor is subject requires it to be built excessively heavy. In the two-stage compressor this inequality of engine excess is not so marked.

The Receiver.—A receiver is a necessity in all compressed-air systems. It is a huge tank with regulating devices to maintain uniform pressure. In a measure it is a power accumulator, if of suitable size; it compensates for the pulsating effect of each stroke of the compressor, and for this purpose should be within 50 feet of the compressor. Another ought to be placed near to the machine drills to reduce friction losses. This will serve also as a drain, if water is present, to catch the water condensing in the pipes.

Further cooling ensues as the compressed air is stored in receiver or pipes, but represents a total loss of power for which there is no compensation. A pound of air not cooled in the compressor, when at 80 lbs. absolute, would radiate 81 B.T.U. in cooling to its initial atmospheric temperature of 60° F., and thus dissipate 63,000 ft.-lbs. of work. Perfect cooling in the cylinder would have saved this amount of steam-power to the motor. 100 cubic feet per minute would thus have saved 16 horse-power, 350 lbs. of steam, and 45 lbs. of coal per hour.

Transmission of Compressed Air.—The air is conveyed to the drill, coal-cutters, hoist, etc., by pipes. With the exception of electricity, no other means of power transmission can compare in efficiency with compressed air. The diameter of the pipe is a matter of the first importance.

The transmission losses appear in two ways: as loss of power and as loss of pressure, or head, indicated by the difference in gauge reading at the ends of the line. There is a distinction between these two losses. The first is the larger, due to cooling of the air during compression, and is unavoidable and not chargeable to transmission. Of the power remaining, some of it is lost by subsequent cooling and some in overcoming the frictional resistances in the pipes. The power depends upon its pressure and its volume. In the process of transmission the pressure is reduced by the frictional losses, but there is a certain compensation from a corresponding increase in the volume. The actual loss of power from this cause is, therefore, slight in ordinary mining conditions.

Frictional Resistance in Pipes.—The loss of pressure, or of head, due to frictional resistances takes place according to laws governing the flow of fluids. If the pipes be short, the velocity will vary inversely as the area, and the frictional loss will be directly proportional to the square of the velocity of the flow. It will be also proportional to the periphery and the length of the conduit. But in long pipes the expression becomes complex. Tables of the loss of pressure by flow in pipes are given by manufacturers, and it will be found therein that air at 32.8 feet

per second loses 8.26 lbs. pressure in a mile of 10-inch pipe, 10.04 lbs. in an 8-inch pipe, and 20.08 lbs. in a 4-inch pipe.

The table below, from the handbook of the Norwalk Iron Company, shows the losses of pressure for given volumes and velocities in pipes 1000 feet long.

q = volume of free air passing per minute at 60 lbs. gauge pressure;
q' = " " " " " " " 80 " " "
p = the loss of pressure-head in pounds per square inch;
v = the velocity in feet per second.

TABLE OF VOLUMES AND PRESSURE-HEAD LOSS, TRANSMITTING FREE AIR THROUGH PIPES 1000 FEET LONG.

Vel.	3 Inches.			4 Inches.			6 Inches.			10 Inches.		
v	p	q	q'	p	q	q'	p	q	q'	p	q	q'
3.28	.046	48	60	.134	86	109	.023	193	244	.014	537	680
6.56	.209	96	121	.152	172	217	.104	386	488	.064	1073	1359
9.84	.488	144	182	.360	258	326	.244	579	633	.145	1610	2039
13.12	.838	193	243	.628	343	436	.419	772	977	.256	2146	2719
16.40	1.317	241	304	.982	429	544	.658	965	1221	.393	2683	3399
19.68	1.808	289	364	1.356	515	653	.904	1158	1466	.542	3220	4079
26.24	3.352	386	486	2.513	687	871	1.670	1544	1954	1.024	4293	5438
32.80	5.270	480	607	3.928	859	1088	2.635	1931	2443	1.573	5367	6798

For any degree of compression p'', other than 60 lbs. gauge, the quantity of free air passing per minute would be obtained by the ratio $74.7 : p'' + 14.7$. This is approximately correct.

EXAMPLE.—Required the volume of free air which at 70 lbs. gauge pressure can be carried by a 6-inch pipe with a friction loss of only 0.658 lbs. per square inch. From the table, 965 cubic feet of free air compressed to 60 lbs. can be carried; hence, with air at 70 lbs. pressure, the volume is $965 \times 84.7 \div 74.7 = 1094$ cubic feet free air.

The Loss of Energy Due to Friction.—The measure of the work possessed by air is represented by that expended in producing the given conditions of temperature, pressure, and volume. Any loss in either of these elements will necessarily diminish its capacity. During compression heat has been abstracted and the capacity for work is measured by the amount of intrinsic

energy remaining. Its value may be obtained from the usual formulæ; the pressure, P, and temperature, τ, at the entrance to the pipe being known. The volume occupied by one pound of air is then ascertained:

$$P_1V_1 = 53.18\tau.$$

Then the relation between the condition of the air at the entrance to the pipe to that at the point of delivery to the motor is ascertained from the formula

$$P_1V_1^{1.408} = P_2V_2^{1.408}.$$

If the temperature has not changed during transmission, the volume and pressure will bear the same relation at the pipe exit as at the entrance, and $P_2V_2 = 53.18\tau_1$. A drop in the pressure ensues, however, due to friction of flow, and this produces a slight reduction in the available energy; that is, P_2 is less by the frictional loss in the preceding table and the volume V_2 has not proportionately increased, making $P_2V_2^{1.408}$ less in value than that of $P_1V_1^{1.408}$ by the amount of energy lost.

The friction in the pipe varies as the velocity and the volume of the air. For a given power the degree of compression may be increased in order to reduce the volume; or the size of the pipe may be increased to reduce the velocity. The efficiency of the system will be increased by either plan. A 6-inch pipe, for example, will carry 800 cu. ft. of free air at 80 lbs. pressure absolute a distance of 5000 feet with a loss of 1 lb. per square inch. For the same volume in a 4½-inch pipe the loss of pressure is 5.362 lbs. per square inch. These pipes would therefore require receiver pressures at the pipe entrance of 81 and 85.3 lbs., respectively, per square inch. The saving in power with the 6-inch pipe over the 4½-inch pipe will therefore be 3 horse-power.

In determining the loss of head due to given conditions, the following formula is available:

$$H = \frac{V^2L}{10{,}000D^5a},$$

in which H = head or difference of pressure required to overcome friction and maintain the flow of the air;

V = volume of compressed air delivered in cubic feet per minute;

L = length of pipe in feet;

D = nominal diameter of pipe in inches;

a = coefficient depending upon the size of the pipe.

VALUES FOR a AND D^5a FOR VARIOUS DIAMETERS OF PIPE.

Nominal D	a	D^5a	Nominal D	a	D^5a
1"	0.35	0.35	3"	0.73	177.4
1¼	0.5	1.525	3½	0.787	413.2
1½	0.662	5.03	4	0.84	860.2
2	0.565	18.08	5	0.934	2918.8
2½	0.65	63.47	6	1.00	7776.0

The Most Economical Size of Pipe.—The friction in properly designed pipe systems is not a serious matter and can be made as small as the most exacting requirements demand, by enlarging the pipe or securing a smooth interior. The capacity of a pipe is somewhat proportional to its cross-sectional area, but is affected by the character of its interior surface and the various couplings used. In calculating the size of pipe required, due regard must be paid to the commercial sizes, for their actual diameter is very different from their nominal diameter. The velocity of the air in the main pipes should not exceed 25 feet per second, and the dimensions should be determined accordingly. At a higher rate the friction becomes excessive and the power lost in overcoming it too large. Though the rate of flow through the pipe is continually increasing from beginning to end, all calculations for frictional loss should bear this in mind, though it may be neglected if the drop in pressure is small.

With due regard to economy in installation, the size of the pipe should therefore be made as large as advisable to reduce friction, but need not be increased beyond that requisite to supply a flow at 25 feet per second. This gives, for ordinary mining practice, a diameter of 4 inches for the mains and not less than

2 inches for the stopes and rooms. A 4-inch pipe with air at 82-lbs. gauge will supply five 3-inch drills 3000 feet away. 100 feet of 1½-inch pipe will serve for only one drill.

Compressed-air Pipes.—The pipes used are steel-riveted or lap-weld, as illustrated in Figs. 131 and 132. The joints should be carefully secured. Means must not be neglected for providing for changes in length due to alternations of temperature. Iron expands 0.000007 its length per 1° F. This allowance is more essential above than below ground; and in shafts where the temperature is inconstant compensation-joints are necessary. At every 300 or 400 feet a copper U tube is attached; its flexibility will allow for contraction or expansion of the pipes. At the Republic mine the brass-lined expansion-joints every 500 feet allow for movements of 12 inches. They rest on gas-pipe rollers. The Chapin iron-mine has expansion-joints at every 680 feet of the 24-inch pipe.

To reduce the frictional losses in the transmission large elbows and bends of long radii should be used. The joints must be made very carefully, to reduce leakage, unless the air-pipes are laid in ventilation-passages of the mine. Leaks when discovered must be plugged. The velocity of escape of compressed air being 20,000 feet or over per minute, a very large loss of fluid will ensue from a neglected leak.

STANDARD STEAM AND EXTRA-STRONG PIPE USED FOR COMPRESSED-AIR HAULAGE PLANTS.

Trade Diameter, Inches.	Cubic Feet in 1 Lineal Foot.	Lineal Feet Necessary to Make 1 Cubic Foot.	Steam.		Extra Strong.		Actual Diameter, Inches.
			Thickness.	Weight per Foot.	Thickness.	Weight per Foot.	
2	.0218	45.41	.15	3.61	.22	5.02	2.067
2½	.0341	29.32	.20	5.74	.28	7.67	2.468
3	.0491	20.36	.21	7.54	.30	10.20	3.067
3½	.0668	15.00	.22	9.00	.32	12.50	3.548
4	.0873	11.52	.23	10.70	.34	15.00	4.026
4½	.1105	9.05	.24	12.30	.35	17.60	4.508
5	.1364	7.33	.25	14.50	.37	20.50	5.045
5½	.1650	6.06	.26	16.40	.40	24.50	5.28
6	.1963	5.10	.28	18.80	.43	28.60	6.065

Sleeve couplings are used in all pipe lines except the wrought iron, which are spiral-riveted or welded tubes. The joints are carefully made, and leaks avoided with the greatest of care. Elbows of as liberal a radius as possible must be provided to have the frictional resistance small.

The Power Value of Compressed Air.—In determining the power value of compressed air it must be remembered that the mean effective pressure of the air is lower than that of steam for a given cut-off and initial pressure.

The mean effective pressure during expansion can be ascertained by substitution in the formula page 316, P being somewhat near the atmospheric pressure. It is less in amount than is the m.e.p. of steam during a similar expansion, as also is its terminal pressure. The work it is capable of equals the work expended upon it if the friction losses be neglected and there were no loss in cooling, for the cycles of changes which the air would experience in expanding are duplicates of those during compression. Much power has, however, been extracted from the air, whose high temperature had been reduced nearly to that of the surrounding atmosphere.

The Work Performed while Expanding from a pressure P_1 and a temperature τ_1, which latter is near that of the atmosphere, to a temperature τ and a pressure P, which is nearly 15 lbs., is

$$W = 183.45(\tau_1 - \tau)w.$$

The final temperature, τ, which is far below that of the atmosphere, produces an intense refrigeration, which is objectionable if moisture is present. The moisture will be frozen in the process and may clog the exhaust-passages. As this is usually the case, the air is reheated so high that after expansion it will still be warmer than the atmosphere. The heat thus expended has the effect of increasing, in proportion to the heat added, the volume, and thereby the amount of work obtainable from the air.

Reheating.—Reheating may be accomplished by direct fire, or by steam, passing through a pipe inside of the air-pipe. Steam thus used gives up all of its latent heat, which can be converted

into an amount of work in the air-engine far exceeding what could be derived from it if used directly in the steam-pump.

A reheater-jacket is sometimes used where other means are not available or desirable. This method would be preferable for compound air-engines which might otherwise require reheating in two stages.

The Efficiency of Compressed Air.—As the efficiency of the driven machine is not over 60 per cent and the efficiency of the compressor is not over 75 per cent, it is evident that, with the losses ensuing in pipes, the aggregate efficiency of the combination does not exceed 25 or 30 per cent of the original motor-power. This, compared with the fuel burning at the boiler, represents an efficiency of about 2½ to 3½ per cent; in other words, the work performed at the drill is equivalent to from 275 to 385 B.T.U. per pound of coal. Notwithstanding this wastefulness, compressed air serves well for many purposes in mining, and will retain its place even against electricity.

The Cummings System of Air Transmission.—This is a closed line of pipes between the compressor and the air-engine. The initial absolute pressure of the air entering the compressor and the final pressure of exhaust from the engine are about 80 lbs., and the terminal pressure in the compressor and the admission pressure into the engine are 150 lbs. This system is used in some pumping plants.

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CHAPTER XI.

PUMPING.

The Water Seepage into Mines.—Water gains entrance into mines by many and devious ways. Into some workings it flows incessantly from some watery stratum, in others the seepage is intermittent. The subterranean current is easily excluded from the mine by the use of a cement lining, or an iron or steel tubing to the shaft, but the seepage accumulates and must be pumped off, unless the workings possess a natural drainage or an easy effluence by adit or tunnel for the upper ground. A gutter at the side of the track, or under the tramway path, with a slope of 1 in 500, readily carries off the water, and not uncommonly delivers it to a small wheel to drive a ventilating-fan. Generally the seepage, following the hydrodynamic law, increases with the depth of the opening, and a very liberal sump is provided for its accumulation. Often one shaft and its workings become, naturally, a sump for the entire district, and drain all the neighboring properties above its level, and this suggests a simple means of keeping one's mine dry. Otherwise, as the amount of water to be encountered is uncertain, provision must be made for the handling of a large volume, according to the history of similar properties. In some coal-mines as much as 4000 gallons of water are raised per ton of coal, and in Colorado 40 tons of water per ton of ore. The Chief of the Bureau of Mines reports that during 1902 1721 pumps delivered to the surface water at the rate of 615,013 gallons per minute from the collieries. The magnitude of such work demands the employment of powerful machinery, and often on a plan too elaborate for the means of the average operator.

Methods of Unwatering Mines.—Some districts were drained by a cooperative scheme with extremely beneficial results. A long tunnel penetrating the country at a level much below the lowest point of exploration drains considerable territory, dispensing with the heavy individual plants and extending the exploration and the productiveness of the mines. Numerous examples of tunnels miles in length may be quoted, some even carrying so much water as to become canals for transportation.

Upon cutting a wet cross-course to the vein, it is a common practice to plaster it up; or, in encountering old workings, to build a brick or stone bulkhead, arched convex towards the water (Fig. 257). To provide means for the escape of the accumulated water, which might otherwise do injury, a cast-iron pipe is built into the dam near its top, and another near the bottom. Either or both may be plugged as required. Similarly, in approaching abandoned works, it is required by law in some States that a bore-hole be kept 30 to 50 feet in advance of the drift, with flank-holes on each side, to guard against dangers from the sudden breaking into the reservoir.

Under certain conditions, in stratified regions, a hole is drilled from the sump down to some permeable stratum, into which the water is discharged.

The water entering a mine at various levels, where economy rather than simplicity is the object, should be led to pumps at the levels where it issues, and not be permitted to find its way to the bottom, to be raised the entire height to the surface.

When the surroundings are such that a tunnel may not be used for the unwatering of the mine, pumping arrangements are indispensable. The earlier forms were crude, the engine being of recent date. Surface waterfalls were employed to operate wheels, which raised bucketfuls from below; or the surface water was arranged to compress air in a reservoir at the surface, from which pipes to the sump conveyed the compressed air, the elastic force of which, in turn, forced the water up to the surface through another pipe. This is a wasteful system and intermittent, but doubtless was cheaper than any other means then available.

The Hydraulic Ram.—At the Comstock mines a special hydraulic ram is used, by which 1800 gallons are pumped from the 2600-foot level to the Sutro tunnel at 1600 feet. The air-pressure in the accumulator is 960 lbs. per square inch, and the pipes at the bottom sustain a pressure of 2000 lbs.

The efficiency of the ram diminishes with the ratio between the quantity of water raised and that used. With a fall of 1 and a lift of 4, the efficiency is 86 per cent; if the lift is ten times the fall, it is 53 per cent; with 1 to 20, it is 17 per cent; and with 1 to 26, it is 0.

Hoisting Water.—Small volumes of water are handled by buckets, obtainable of any size, and with a capacity up to 200 gallons (Fig. 130). At the bottom is an inlet-valve by which the tub is quickly filled as it sinks into the sump; it is then hauled up, its valve closes, and at the surface it is discharged by being brought down on a pin which again opens the valve. In some mines the water-bucket is attached underneath the cage, and travels continually with it. Bailing-tanks holding 500 to 1000 gallons, balanced in pairs in one compartment, are hoisted by a special drum. Slopes are equipped with similarly valved skips, the emptying being done from the mouth, as with ore.

FIG. 130.—A Water-bucket.

Bailing-tanks should always be in readiness for immediate service at every mine operated through shafts or inclines, to relieve the pump on emergency. Bailing is not economical, because of the great dead weight of tank and rope compared with the weight of water hoisted. They are adapted to a variable inflow and give an equal, though low, efficiency with all volumes of water.

Mine-pumps.—The design of mine-pump plants requires the greatest possible reliability against breakdowns, the greatest

possible facility for making repairs, and, if practicable, the highest mechanical efficiency at the normal rate of flow.

The pumps used in mining work include reciprocating pumps driven by rods; steam-, air-, or water-engines; rotary pumps operated by electricity, water- or steam-turbines, oil-engines, or belts; pulsometers, injectors, etc., employing steam; and siphons.

The reciprocating pumps include the plunger, piston, and bucket. Of these the oldest is the lift-bucket, which, however, affords no means of repairing the barrels and packings, which wear rapidly with sandy waters. The piston-pumps are suitable for low pressures, but cannot be repaired as readily as the plunger-pumps, whose packing and stuffing-boxes are outside and can be replaced without stopping the pump. They may be driven by rods from a motor at the surface, or by a motor directly connected, receiving power through wires or pipes. The rod-driven pumps have many disadvantages; nevertheless, even with the competition from steam and air-driven motors, they are much used.

Belts or hemp rope may also be used to drive pumps on a separate foundation. They are called power-driven pumps, having the additional advantage that they may be located remote from the power source. Of these the rotary pumps and centrifugals have a capacity which can be economically varied within very narrow limits.

Injectors, pulsometers, ejectors, etc., are to be regarded as temporary substitutes only during the repairs to the main plant.

Mine Rod-pumps.—The rod-driven pumps are of two types. One, the lift-pump, is a single line of rods, which reciprocates in a straight line of pipe. The lift-pump is a direct-driven pump, with vertical single-acting engine by which the water and the rod are lifted on the up-stroke, while the weight of the latter carries it down. The other, the Cornish system, is an extended rod operating outside of the column-pipe upon a number of plunger-pumps along the line, with a lift section at the bottom. In the Cornish pump the water is forced up on the down-stroke by the weight of the rods. The engine at the surface raises the rods

and plungers besides lifting the water in the suction-lift. The Cornish system employs a rotary engine, making it possible to utilize the expansive force of the steam.

Pump-pipes.—The pipe through which the water flows is variously termed as a stand-pipe, column-pipe, and lift. It is of a diameter commonly 10 inches, often as much as 20, and extends from bottom to top. It is of cast iron, wrought-iron lap-weld steel or spiral riveted, or welded, in standard lengths of 5 to 20 feet. The cast-iron pipe, having a smooth interior and uniform diameter throughout, is preferable and more convenient than the riveted pipe (Fig. 131) or the lap-weld iron (Fig. 132); but as it represents too much dead-weight for the strength, its days of utility are nearing an end. The ideal pipe is of steel, which gives the lightest, strongest, and most durable tubing; this may be had in four grades, light to extra heavy. It is made of sheet steel spirally laid, riveted at the overlapping joints or cold hammer-welded. The pipes are united by bolting together at the flanges, which are riveted, screwed, or locked on the pipe (Fig. 132); or, preferably, they are coupled on the hub-and-spigot plan of sleeve (Fig. 133). This is a double socket, into which the pipe is slipped, "oakumed," and leaded from each side, as shown. For this joint the pipes have expanded ends.

FIG. 131.—Spiral Pipe.

A water-tight joint is secured by placing rubber, leather, lead, or, best of all, corrugated copper gaskets between the flanges, which are then bolted together while lowering. Spence's metal, used as a calker, offers an excellent joint, is cheaper than lead, and ought to be better known.

The pipes last from fifteen to twenty years unless the water is corrosive, in which case gun-metal is used. If the water is very

Fig. 132.—Pipe-joint.

bad, wooden pipes are made by hollowing the trees, fitting the joints, tarring them, and strengthening by wrought-iron bands at every 3 to 6 feet. In many mines recourse has been had to

Fig. 133.—Pipe-joints

these as the only stand-pipe that will last over six weeks.

In calculating the flow of water through pipes, the effect of entrance head, the friction factor, the effect of bends, elbows, tees, valves, gates, etc., as well as the condition of pipe-joints and the internal surface of the pipe, should enter into the calculations; otherwise the results obtained are mere approximations.

Care should be taken to give independent support to the water-pipe lines to prevent motion, to relieve the lower sections of the pressure from above, and to avoid the evil consequences of vibration. They are always stayed laterally to keep them in line. In no case should the supports be rigidly connected, preventing expansion or contraction. All bends and elbows should be supported, as the curve of the pipe receives excessive pressure and is also of weak resistance. All the bands are made of cast-

ings, and the long bands may be of riveted pipe. The acute-angular turns are made by inserting thick wedge-gaskets.

The size of the pipe should be as large as possible, that the velocity of flow may be maintained with little resistance. The suction-pipe should be short and more liberal in diameter than the delivery. The velocity of water in the pipe differs little from the velocity of the plunger or piston. Water-hammer and serious consequences would be the result, if water were allowed to flow intermittently through pipes at more than 300 feet per minute.

Owing to the corrosive action of most waters and their solid contents, the pipes are made as thick as consistent with other conditions in the problem. The varying pressures, the acidity of water, the water-hammer, and the weight of the column which is supported, are elements which increase the thickness of the pipe.

Valves on Pump-pipes.—Air-chambers are supplied at convenient points of long hydraulic pipe, if no provision is made for the insertion of relief-valves or pressure-regulators. The air-chambers should be made air-tight and coated with heavy asphalt for heavy pressures. Relief-valves are spring-loaded or weighted pistons, or valves set to open at a given pressure. The former are more sensitive than the latter. Ample lift should be given to allow of a prompt discharge of air or of water to relieve the excess pressure promptly.

Automatic air-valves on hydraulic pipe lines are sometimes used, constructed so that they will close by the combined effect of buoyancy and the velocity head of the water. Some air-valves are provided to admit air and prevent a collapse of the pipe from atmospheric pressure when the pipe is emptied of water. They do not operate, however, to let air into the pipe until the pressure falls very low. The pipes should be located in the up-cast shaft compartments, the steam and reheated air-pipes being protected also against radiation. Steam-pipes and air-pipes are provided with traps at low points for drains.

Check-valves must not be neglected where the conditions are such that a stoppage in the pump may cause a reverse flow of

the water with consequent danger to machinery. All valves and gates and water-pipes should close tightly and without shock.

Pump-valves.—The valves for pumps are hinged, called clack-valves, straight lift-valves, rising vertically from their seats, and flexible valves, which alter their form on opening. Direct-driven pumps employ straight lift-valves of rubber or vulcanite resting on a grating. The requirements of a good valve are that it should close promptly and perfectly on its seat, open easily, and remain with a minimum of pressure. It should not present much resistance to the flow or divert the current from a straight line. It should be simple and accessible.

If the waters carry material in suspension, the whole valve should be made of some elastic material in order that the particles lodging on the seat shall not spoil the band. Leather cannot be used if the water is acid. Rubber compositions are employed for hot water. Hinged valves are more liable to leak than straight lift-valves. The use of wooden seats for metal-faced valves is objectionable because they leak, although they are more desirable than rubber or leather. For acid waters the seats are usually of brass screwed into position, and thus easily removed when too far gone from corrosion. The straight-line lift-valve (Fig 141) of rubber composition rising from a grating and held in brass cages is used for pressures up to 500 lbs. to the square inch, but above that the discs are of metal with leather facing. Flexible valves are generally of rubber, suitable only for moderate lifts, and round in form, or rectangular with all of the corners trimmed.

The area of the valves is large to allow of a free flow of the water, but increasing the area of the valve increases the pressure upon its surface and requires a heavy, thick valve. Hence it is more desirable to use a large number of small valves than to employ few of large area. On the other hand, it is desirable to restrict the lift of the valves as much as possible and to reduce the velocity and consequent resistance of the flow past the valve to a minimum. Enlargement of the area increases the risk of leakage and also the percentage of slip. Again, increasing the

area of the valve increases the width of the bearing and its seat. This makes the difference between the lower area of the valve subject to pump pressure and the upper area of the valve subject to the resistance in the pipe so great that an excessive over-pressure will be required to raise it.

The Working-barrel.—At the lower end of the stand-pipe a 12-foot length of cast iron constitutes the working-barrel, in which oscillates a piston carrying an upward-opening valve, similar to that at the lower end of the barrel (Fig. 136). For acidulous waters the barrel is bushed with gun-metal. It should be thick, to admit of being bored out several times, as it is rapidly cut away by the gritty waters during sinking.

The working-barrel can never be more than 28 feet from the sump-level; in mountainous districts still less; at 5600 feet altitude, 23 feet; and at 10,000 feet, 18 feet. Usually the working-barrel and suction-pipe are suspended by chains from two stulls resting in the cribbing, and the stand-pipe supported at intervals by stout reachers.

The Suction Length.—Below the barrel is a length of pipe or flexible hose dipping into the sump and receiving the water through a perforated strainer. During sinking this suction-pipe must follow the lowering of the sump. During the blast it is raised for each shot or boarded over.

The strainer prevents sand and gravel entering the cylinder. The suction-lift must be as short as possible and as large as admissible. As the water-level lowers during the sinking operations in a mine, a stationary pump should be provided with a wire-wound flexible hose below its suction-lift, dipping into the sump, or a telescopic joint allowing for a supply of 10 feet. The hose is preferable because it can be adjusted to the bottom of the shaft. Owing to injury and hard usage it is usually covered with canvas and wire-wound.

When the water-level has receded below the allowable suction height of the pump a length of pipe must be added above the pump which is then lowered to a degree determined by fixed conditions. Under such circumstances the working-pump

is suspended by chains from two stulls resting in the cribbing, the stand-pipe above the pump being independently supported at intervals.

The Lift-pump.—The piston, or "bucket," is attached by an iron fork (Fig. 134) to a wooden rod, 4 or 5 inches square, ex-

FIG. 134.—Single-acting Steam Lift-pump.

tending up through the pipe to the surface, where it is connected either with one end of a walking-beam or to the piston of a single-acting engine. As it receives a tensile strain, the joints are scarfed and strapped, or, if the ends are flushed, two continuous lines of strap-iron, breaking joints, are bolted together through the rod. The latter plan reduces the breakage and the number of stoppages for repairs. A 4-inch rod is large enough for a 12-inch

pipe, and a 5-inchrod, properly spliced and strapped, for a 13-inch to 16-inch delivery. The size of the straps is easily calculated. The area of each one, a, is $0.000075d^2D$. A 200-foot pump-rod requires two straps $4\times\frac{1}{8}$ or $3\times\frac{1}{6}$ for a 10-inch pipe.

On the down-stroke the rod falls through the column of water, while the valve in its piston opens and the clack of the working-barrel closes. Returning, the valve's action is reversed, water rises from the sump into the working-barrel, and all that above the piston is lifted a distance equal to the stroke, and a column of water simultaneously discharged at the surface.

At the surface the column-pipe terminates in an elbow discharge or in a laundry box and trough, the pump-rod continuing up to the framing. The mechanism by which the motion is communicated to it is simple. A stout frame, with two samson posts, supports a walking-beam receiving its oscillatory motion

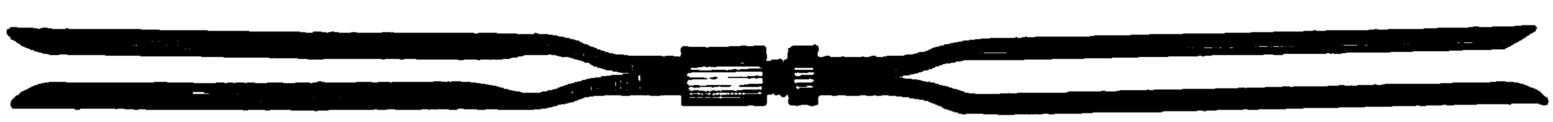

FIG. 135.—Connection for Lift-pump Rod.

from a pitman actuated by a crank-arm adjustable to a 1-, 2-, or 3-foot radius, giving strokes of double this length, at the opposite end, to the pump-rod, which requires little force besides its own weight. The arm is on a shaft turned from the engine by cog, geared 1 to 6 or 7, giving 12 to 20 strokes per minute to the rod. The ironwork of this frame, inclusive of cogs, pulley, and castings, will cost about $250; the woodwork, including a 24-ft.×15 inches square walking-beam, about $125.

Lift-pump Plunger.—The valves are made of several thicknesses of oak-tanned leather cut into discs, tacked together, and slipping easily on a grid at the top of a cast-iron cellular ring-bucket. A perforated cast-iron guard on the grid limits the rising of the valve as the water passes through the bucket. The cellular-ring bucket casting is all there is of the piston, which slides freely in the barrel, and has no other packing than that offered by the leather discs forming the valve and which are cut larger than the cylinder. The rapid movement, the wear, par-

ticularly during sinking, and the heavy pressure upon these valves consume a set every two weeks, and, though substitutes have been suggested, such as flexible brass or gutta-percha plates, they are not more durable, nor have the brass balls or conical poppets had any marked success. The valves are repaired or replaced by removing a bolted door-plate in the barrel opposite them (Fig. 136).

The Direct Lift-pump.—The framework just described is crude. A simpler connection is a vertical steam-cylinder placed over the shaft wi h its piston bolted to a fork on the rod This dispenses with framing (Fig. 134) beyond that necessary for foundation. The piston may be single-acting, though frequently the steam is admitted on both sides These cylinders are easily set, their cost is low, and their maintenance is small. Cylinders of $12'' \times 30''$ are operated at a rate of 24 double strokes per minute. The Bull pump is an example of this type, well known in collieries. It is single-acting, and is as large as 60 inches in diameter, making 6 to 8 strokes of 10 feet each.

It is not certain that this form of pump gives a higher duty than the drive-rod pump. Except for increasing the speed there is no occasion for using steam on the down-stroke. An economic degree of expansion on the up-stroke is not possible. The main objection to its more general adoption is the large area of

FIG. 136.—Drive-rod Plunger.

shaft-mouth it covers. Besides, to lengthen or repair the rod or column-pipe the cylinder must be displaced, or the additions are made below; either is slow. This pump cannot be used in slopes; the irregular wear of the cylinder on one side cannot be compensated for, nor the friction of the rod in the pipe counteracted.

The Capacity of Lift-pumps.—If d be the internal diameter of the pipe in inches, and L the stroke in feet, the discharge in gallons with each upstroke is $0.0408d^2L$; and the work done per minute, in foot-pounds, exclusive of resistance in the cog-gear and the mechanism for transmitting the power, is $0.34272d^2LND$, where N is the number of double strokes per minute, and D the height of the water-column in feet. The direct-connection lift-pump wastes less power in friction and has an efficiency of about 85 per cent. Its least working steam-pressure is represented by this equation:

$$k^2p=0.345d^2D \quad \text{and} \quad k^2p=0.551(0.7854d^2-a)D.$$

With a moderate steam pressure the ratio between k and d becomes large if the shaft is deep, and the area of the wooden rod, a, must be large enough to support the load. When the depth of the shaft has reached 250 feet the lift-pump is no longer practicable, and must be altered to a single discharge-force, or replaced by the more economic continuous-flow steam-pump.

Force-pumps—Rods.—The pump-rod of the force-pump, unlike that of the lift-pump, works outside, not inside, of the stand-pipe. Its lower end is bolted to one end of a cast-iron H chamber, the other stem of which carries a long working-barrel, into which plays a solid plunger-piston (Fig. 136), instead of the bucket-valve of the lift-pump. Below the stem carrying the stand-pipe is the suction-pipe, and in it is an upward-lift clack-valve. Then during the up-stroke the lower clack opens, while water rises through it into the working-barrel; as the plunger falls the water is driven through the upper valve against the column of water in the stand-pipe. As sinking progresses, suction lengths are

added at the bottom until the sump is lowered beyond the suction distance, then the pump is lowered, while additional lengths of pipe are attached between it and the H piece at the discharge-station above, until a lift of 300 feet has been reached; then the suction-barrel is removed, to do service similarly for the lower lifts, and is replaced by another H piece.

The Cornish Pump.—The Cornish system is a development of the rod-driven pump by which the lifting section is added to a series of force-pumps in a continuous line, the water being driven by stages to the surface (Fig. 138). The engine raises the rod, and water is sucked up into the working-barrel at the bottom, while that above the bucket is lifted to the first station above. Here the plunger is drawing water up through the lower clack into its barrel, all the other plungers doing the same. At the end of the stroke (6 to 10 feet) occurs a slight halt, incidental to the change of direction. The rod falls by reason of its own weight, and each plunger closes the lower clack in its H piece, opens the upper one, and forces the water out of its working-barrel, driving at the same time the entire lift-column an amount equal to that of the stroke. At the surface a volume of water is discharged on the down-stroke.

The Cornish Rod.—The pump-rod extends down the shaft, and terminates at the bottom in the piston of the bucket-lift. At intervals are offsets, to which are bolted the rods carrying external plungers, not pistons, to move the water, and are therefore more easily packed and admit of pumping against higher heads, and remaining tight much longer, than piston-pumps, being much less subject to wear. They are made of cast iron, though brass is a better material, having also less friction. The stuffing-boxes for packing the plunger are separate from and built to top of the pump-level. The usual packing is of square braids of hemp, flax, or cotton soaked in tallow or a mixture of tallow with beeswax. Flax is the cheapest, but hemp is more durable. Both of them offer greater friction than leather as a packing. The latter is used in all pistons and bucket-pumps.

Each plunger reciprocates within its working-barrel, which

is one leg of an H piece (Fig. 137). At every station is located an H piece having at the top and the bottom a hinged valve opening upward. The working-barrel is about 15 feet long.

Excepting the short suction-lift length at the bottom, the column-pipe is in a continuous line, broken only for the introduction of H pieces, or tanks, at the stations.

The lifts are rarely over 300 feet or less than 150 feet in height. As the stations require heavy timbering, especially if tanks are used, their dimensions must be large and their supports solid and independent of the shaft-timbers. They are not any more numerous than the circumstances demand. The greater the number the greater the speed of pump and the smaller the diameter of the pipe will be for a given first cost, exclusive of that of the tank-chambers.

The pump-rod is composed of sections joined at their ends by iron straps to bear the continual reversal of direction of stress. Their length is as great as convenient to handle, and their section as large as necessary to resist the tensile and compressive forces to which they are subject. On the up-stroke all sections are in tension. On the down-stroke the net stress is the resultant of the compression from forcing the water and the tension of the pendent weight below the section, modified by the inertia of the attachments and counterweights. The counterbalances reduce the tension and increase the compression by an equal amount.

The aggregate weight of the rod is greater than that of the water to be moved. Its cross-section is smallest at the bottom, increasing to the top with the increase of tension upon a section from the pendent weight below.

To prevent accident from breakage or buckling of the rod, stout stulls are laid across the shaft at intervals close to the rod to catch corresponding "wings" of heavy timber clamped by iron collars on the rod. The stulls are called "guides," or "stays."

In a deep mine requiring many superposed sets of pumps on the same rod the stroke of the lowest length is shorter than that of the topmost plunger. This may require an increasing area

of the plungers from bottom to top. Usually, however, they are the same. It not infrequently happens that there is a differ-

FIG. 137.—An H-piece Cylinder of a Cornish Pump.

ent displacement of the pumps, which may cause pounding or other troubles. A float provided at each station to tap the pipe

into the station tank when it happens to be drained too low would maintain a supply of water and prevent overflow of one tank and the draining of another.

Elastic bumpers are also needed to break the force of the blow which might occur with a stroke greater than the intended limit through a variation of the steam pressure or neglect in regulation of the water flow. As an instance of the size of a rod—that of the Maira, 2300 feet deep—we find the first 780 feet down was $16'' \times 32''$, tapering to $12'' \times 24''$; at 864 feet it was 16 feet square; it tapered to 14 inches square at 964 feet; thence it was 13 inches and at the bottom 12 inches. In well-ventilated shafts wood is the preferable material for rods, neither wrought-iron rods nor wire rope having the requisite resilience.

In vertical shafts the rods fall freely by their own weight; in slopes they rest on friction-rollers, placed about 30 feet apart. When iron ropes are used instead of wooden rods, sheaves support them. Changes in the slope may be provided for by the use of the rocking-arm. A chamber is cut in the shaft at the angle, in which is firmly set a frame on which swings a V bob by a hinge-pin at the apex. To the two arms of the angle the inclined rods are attached. While this arrangement is not desirable, because of the expense and the loss of power, still it is the best to be had when slopes are sunk on contorted veins. The application of the Cornish system to inclines is attended with many drawbacks.

In mines utilizing the pump-rod for a man-engine additional counterbalance weights are connected at intervals down the shaft (Fig. 138). Sometimes two lines of rods are used in a shaft, working two pumps from the same bob, in which case no counterbalance is needed.

The ironwork of the rod should be protected against rust, particularly the joints and the inside of the hollow iron pump-rods. Pickling in acid will remove the rust, after which a coating with warm oil and red lead is recommended.

The Balance-bob.—At the surface the rod is directly connected to a pin at one apex of a king-post balance-bob. If it be of iron instead of wood, a link is used for a flexible connection.

The horizontal beam of the bob is about 25 feet long, with a saddle and axle underneath near the centre. From the upper end of the king-post, which is 8 feet high, a connecting-rod leads to the engine. Besides the braces on each side down to the beam there are two tie-rods, taking with the braces the tensile and compressive stresses. All the members of the frame are of wood or iron, in iron shoe-castings at the ends. The frame stands vertically in a pit dug alongside of the shaft, 8 to 10 feet deep (Figs. 35 and 138). The rocking motion is communicated to the bob by a connecting-rod, or "pitman," operated from a wheel geared to the fly-wheel shaft of the engine, the work of which, during the up- and down-strokes, is somewhat equalized by the bob and its counterpoise. The crank-pin can be set to different radii to alter the pump-stroke, increasing the leverage of the engine for the greater depth of pumping. The third apex is occupied by a box full of iron and boulders, to counterbalance the excessive weight of the rod; for it will be found that the weight of the long column-rod of a strength requisite to force the water up is much greater than that of the water pumped. A certain pump, raising 440 gallons 1690 feet by six lifts in a 22-inch pipe, has a balancing weight of 33 tons on the bob.

The Cornish Engine.—When once placed and its speed regulated, the Cornish pump gives little trouble. It is the most reliable and also the most expensive pump in use. It has numerous advocates as against the steam-pumps; but in transplanting the system to America its main redeeming feature—the cataract-engine—was discarded, while persistently clinging to the worst—the cumbrous bob and rod. When the vertical direct-acting engine was introduced it was thought to be a great improvement, because of the suppression of the heavy bob; but it was soon discovered to be a mistake, and the beam was quickly reestablished, with a Corliss horizontal engine as the motor. An engine, boiler, and fittings complete, with three 15-inch plungers and one 16-inch lift-pipe, etc., etc., for a 600-foot shaft, weighed 650,000 lbs., had a capacity of 800 gallons, and cost in place $54,000.

FIG. 138.—The Cornish Pump.

stroke an acceleration before
the water had fully entered the

cylinder. A shock ensues which is detrimental to the rod and valves. If continued, a vibration or churning occurs with the continual pound of descending plunger upon the rising water column. The speed of the pump should be reduced or the engine stopped entirely for a time. If after starting it is again set up the fault may be with the valves giving an obstructed flow.

The valves should afford unobstructed passage to the water in one direction, and close perfectly in the other—two antagonistic conditions which can be attained only partially. The strain on the valves is enormous, and if tried too hard they become weak, do not work properly at either stroke, and lose water by their "slip." If the valves are found to be free and the vibration still continues, the joints of the rod are loose or the bob requires resetting. Increasing the mass of the rod is sometimes, though not always, a remedy. This vibration must be particularly guarded against if the rod is also to be utilized for a man-engine. For convenience in repairing, the pumping compartment should be provided with a ladder-way, with plats and chambers.

Cushier's Double-acting Drive-rod System.—The use of a double-acting pump, retaining therewith the advantages of the Cornish, would save space in the shaft, the pipes for a continuous discharge occupying less than one fourth the area of a single discharge-pipe and its rod.

Cushier's system of pumps for deep mines consists in having sets of two pumps, each working in concert, one above the other, the suction- and discharge-pipes being common to both pumps.

The pumps are placed at intervals of about 200 feet in the shaft, the power being transmitted directly through the centre of the plungers. The connection with each other and to the motive power is effected by means of a steel-wire cable encased in wood, preserving it from external wear as well as from rust. This cable is fastened to, and its length regulated by, shackle-bolts.

The plunger of the lower pump in a set is double in area that of the upper one, so that in working on the upper stroke

one half the water raised fills the chamber of the upper pump, the other half being forced out through the discharge-pipe on the down-stroke; the upper pump-plunger forces out, in its turn, the water in the chamber, thereby causing a continuous delivery.

This form of pump can be worked at any angle, to any depth, and is almost perfectly balanced. The last-named advantage enables it to be connected with, and worked by, a direct-acting steam-cylinder, and thus does away with the complicated gear and bob of the Cornish.

Sinking-pumps.—During operations of shaft-sinking a double-acting steam-pump may be suspended vertically from some support by a chain to a bale, the suction-inlet being at the bottom of the pump. By this suspension it can be accommodated to a varying water-level. A centre-packed plunger is directly connected with the steam-piston, from which it receives its power, as in the horizontal pump (see Figs. 140 and 144). The sinking-pumps of the Cameron, Knowles, and Worthington companies are of similar pattern. The valves are absolutely positive, and are protected by a cast-iron shield serving as a yoke between the steam and water ends, while those in the steam end are cushioned to regulate the strokes. Hand-hole plates, with hinged bolts, allow of easy repair of the valves and the shoes and dogs (at the left of the pump, Fig. 139) of easy and simple means of support.

Sometimes the sinking-pumps are attached to a sinking-frame guided in the shaft. The frame and pump are raised as occasion requires by chain-blocks, winches, or special hoist at the surface.

The Reciprocating Pumps.—A plunger driven by steam, air, or water pressure in a cylinder directly connected with the water-piston or plunger, the two being rigidly coupled and having a common stroke (Fig. 141), constitutes a reciprocating pump. It is always double-acting and may be single or duplex, and is often compounded at the motor end, with, in some cases, a condensing connection.

Fig. 130.—A Sinking pump.   Fig. 140.—Section of a Sinking pump.

FIG. 141.—A Plunger-pump.

The column-pipes are smaller in diameter than for the Cornish system and the piston speed is greater, but, being of the non-rotary type, no advantage can be taken of the expansion to any great degree. The water end may have a piston (Fig. 142) or a plunger (Fig. 141). The latter is used for the higher heads of lift.

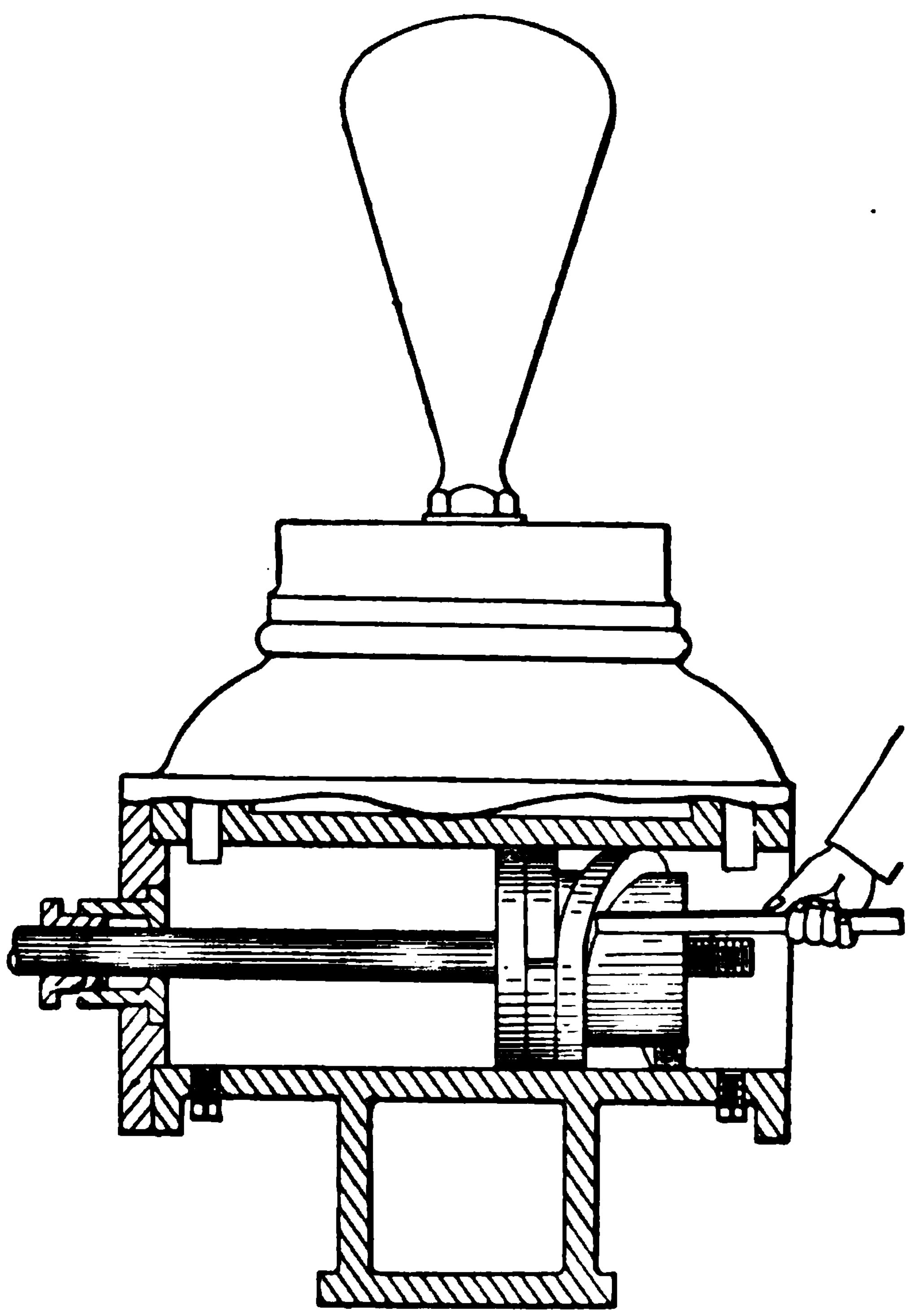

FIG. 142.—Packing a Follower Piston.

The steam-pump dispenses with the cumbrous gear, bob, and rod, having instead a small, well-lagged steam-pipe, conveying the power down from a surface boiler. Its construction is similar to that of the air-compressor (Fig. 126), consisting of a steam-cylinder in which a piston oscillates, moving its own steam-valves by rockers without the aid of any rotary appliances; at the same time it reciprocates a solid plunger centrally in a water-

cylinder, at each end of which is a set of double-beat valves of appropriate construction, open only so long as the water is being forced, and closed with the aid of springs. One valve is removed in the figure.

Pump-valves.—The inlet-valves are opened back of the water-piston by the pressure of the atmosphere, which forces water into the space thus provided. The water is expelled through valves into the discharge-pipe on the return-stroke. At the other end of the water-cylinder similar conditions prevail, the delivery-valves being raised against the action of the springs which hold them to their seat (Fig. 141), and the exhaust-valves being raised by atmospheric action.

The steam-valves in duplex pumps are like those in the steam end of engines, but have no lap. They are provided, however, with considerable allowance for lost motion, which prevents one port from opening as soon as the other port has been closed, at the end of the stroke of the piston. The valve-stem does not engage the valves rigidly, and hence imparts no motion to the valve until the piston operating it has nearly completed its stroke. The amount of lost motion is about one third the width of the steam-port. If the piston does not make the full stroke, the valve-chest cover must be removed and the lost motion increased.

Single-cylinder Pumps are often preferred to duplex pumps, because they deliver more water in proportion to their size and the amount of steam used. There is no interference in the action of the piston by the tightness of the stuffing-box on the other side, as in the duplex; hence there is no tendency to reduce its piston speed. In the single-cylinder pump the piston never starts on its return until it has travelled its full length of stroke. The steam economy is due to a smaller radiating surface for a given capacity and the diminished clearance. The valve-gears of the various types are automatic and give little trouble, and are not complex in construction. They usually consist of some auxiliary valve, which opens and closes the exhaust and steam admission for the main valve. The former is moved by the

piston-rod, and the latter controls the movement of the piston. The piston of the single-cylinder pump always makes a full stroke.

The Plunger-pump. — Where great pressure or the gritty nature of the water renders the use of the single piston undesirable, the water-cylinder is divided in the centre, and a pair of plungers, discharging alternately, work in the opposite ends, and are connected with yokes and heavy outside rods to the steam piston-rod (Fig. 145). This arrangement of external stuffing-boxes permits of instant detection of leaks. Strictly speaking, the combination is a pumping-engine; but this term is customarily applied only to the double- and triple-expansion engines used for city supply.

The pump-cylinders are best made with brass linings. The piston is packed with hydraulic packing (Fig. 142), hemp soaked in Albany compound for cold water, or square-braided cotton mixed with plumbage for hot water. Such packing is held in place by the follower. Rings of square rubber or double cup-leathers are also sometimes used.

The plunger is usually hollow and of such thickness that it will be of the same weight as an equal volume of water.

An air-chamber is used on pumps to maintain an equal discharge. It is placed at the highest point on the discharge and permits the air to rise to form a cushion which maintains uniform pressure upon the column of rising water. Its volume is about twice the volume of the water-cylinder of duplex pumps.

A vacuum-chamber on the pump will keep the water in full motion, and stop it gradually and easily. It may be placed at the side or the end of the pump. Its action is practically the reverse of that of the air-chamber and facilitates the changing of continuous into intermittent motion.

The suction-hose is connected under the inlet-chamber, and the discharge-pipe to the surface on top or at one side of the outlet-chamber. The water-passages are short and very direct; the valves should be large, move quickly, and close tightly, that little loss be experienced; otherwise the effective and suction powers are both reduced.

The principal difficulty is with the disposal of the exhaust. If turned off into the sump, as in Fig. 143, the temperature of the mine is raised, ventilation is injured, and the timbers ruined; if carried to the surface, the condensation in the pipe gives trouble. The best remedy is to use a condenser, which reduces the back pressure and increases the efficiency. Jacketing the cylinders materially contributes to economy of steam consumption.

The condensation of the steam can be carried into the suction-pipe, as is the universal practice, being arranged to enter the pipe in a direction parallel to the flow of the water. This enables it to act like an injector, and aid in accelerating the lift of the water (Fig. 143).

The Comparative Merits of the Steam-pump.—The direct-acting steam-pump does not equal the Cornish cataract pumping-engine in fuel economy. A fly-wheel is necessary in order to secure the full benefits of a high degree of expansion, which, as has been seen in Chapter V, is not feasible in one cylinder, because the resistance (the weight of water forced up) is constant throughout every stroke. A fly-wheel would distribute the steam-power excess of the first part of the stroke to the latter. By the use of the compound cylinders the saving of steam-power may be fully 10 per cent. The ratio of expansion is rarely over 3. The Deane compound pump with externally packed pistons is illustrated in Fig. 145.

Direct-acting pumps are generally made as light as possible, but it is not practicable to employ the steam expansively. The steam is admitted during the full stroke of the plunger, as is revealed in the indicator card, Chapter V, showing the constant pressure required for the uniform water resistance. The pressure may be changed within limits of the boiler supply.

For general purposes these direct-acting force-pumps are in universal use. Their chief feature is their comparatively high efficiency at any speed, slow or fast; they are capable of quick adjustment in speed and discharge, as emergency demands; but they require close watching, especially where the water is "quick," or they may be drowned. The Cornish pump does

not admit of variations in its rate: during summer and winter it is run only a few hours through the shift to empty the sump which has been filling overnight. The small cost, great simplicity, and ease of repairs give the steam-pump an important advantage. A plant with boiler, pipe, and fittings, complete, can be installed for less than one fifth that of a Cornish outfit. One for 850 gallons per minute, 400 feet, cost $15,000 in place. These pumps have timber foundations, in a large, well-

FIG. 145.—Deane Compound Pump with Externally Packed Pistons.

timbered excavation alongside of the shaft and near the sump level, which must practically be invariable. They are useless during sinking without a sinking-pump to deliver to their tanks. In coal-mines, and where the machinery can be established for a permanent bed, these pumps have no rival (especially if compounded); whereas in vein-mining the pumping apparatus, and indeed all of the machinery, is under process of continual extension. For this reason in metal-mines the choice must be between a set of relays of direct-acting pumps at each 200 or 300 feet with a sinking-pump at the bottom (Fig. 144), and the Cornish pump with its several force-stations and its bottom-lift (Fig. 138).

The Capacity of a Pump.—In determining the horse-power and the amount of steam necessary to operate the pump running at full capacity, the calculation will be made in the same manner as for steam-engines, bearing in mind that the steam

cannot be used expansively. The volume of steam consumed per minute can readily be determined by the continued product of the area of the steam-piston, the number of strokes made by all of the plungers in a minute, and the length of each stroke. This product, reduced to pounds by reference to the steam-tables and divided by 34.5, will determine the boiler horse-power of its boiler. Pumps usually have boilers independent of the other steam-consumers.

Steam-pumps are very wasteful of steam, the simple direct-acting type using at least 100 lbs. per horse-power per hour; the duplex may consume as much as 300 lbs. of steam; the compound non-condensing may not use more than 70 lbs., and the compound condensing but 40 lbs. per horse-power hour.

Let W = weight of steam required by the pump per hour, and
w = " " 1 cu. ft. steam at the pressure in the cylinder.

Then k, s, and N being the diameter, stroke, and number of strokes per minute, $W = 0.0341k^2sNw$.

When it is found that a boiler pressure exceeding 100 lbs. per square inch is necessary for the given lift of water, either the pump or the boiler pressure must be increased, or the height of the lift reduced. With a given steam-pump and boiler pressure it would be necessary to employ several pumps in relays, each pump delivering to the tank above and thence to the surface. In Fig. 144 is shown the disposition and arrangement of such a set.

The Duty of the Pump.—The calculation of the steam and fuel economy is easily made; the necessary elements are few in number. The standard of comparison of the work of a pump is its duty—the number of foot-pounds of work actually performed per bushel (80 lbs.) or per 100 lbs. of coal. The combustion of one pound of anthracite gives sufficient heat, theoretically, to do 12 million foot-pounds of work. The ratio between the work actually done and the power at the steam end measures its efficiency, to the consideration of which, in and about mines, insufficient attention has been given notwithstanding its pecuni-

ary importance. The duty of a small pump is from 7 to 15 million foot-pounds per 100 lbs. coal; a compound pump gives from 15 to 30 millions; while the higher types of pumping-engines furnish from 30 to 100 million dynamic units, corresponding to the consumption per hourly horse-power of 28 to 13, 13 to 6.6, and 6.6 to 2 lbs. coal respectively. The consumption of coal per hourly horse-power equals 198 divided by the duty (in millions). A recent report of a Worthington engine having a capacity of over 1000 gallons per minute, against an equivalent of 2000 feet head of water, showed a duty of 184 ft.-lbs. per thermal unit, or 158,-000,000 dynamic units per 100 lbs. of coal.

To illustrate the influence of compounding and jacketing the steam-cylinder, and of condensing the exhaust, upon the coal bills, two examples will suffice. As has been stated, many of the collieries pump 4000 gallons of water per ton of coal hoisted. To raise this only 300 feet requires the consumption, theoretically, of 336 lbs. anthracite for a daily output of 400 tons. If the duty be 90 million foot-pounds, as in Cornwall, or 20 million, as with our average duplex compound pumps, the aggregate yearly consumption is 675 and 3005 tons respectively of anthracite, or of 900 and 4000 tons of lignite.

But duty is not the sole feature of a piece of machinery: the repairs and lubricant accounts and the durability of the plant are not to be overlooked; for the indicator card is a less valuable guide than are the coal, oil, and packing bills. Moreover, the inconveniences arising from the use of steam underground and those of the occupation of a shaft compartment by rods, catches, etc., must receive attention. The cost of pumping per million foot-pounds is not far from 1.6 cents with the direct-acting pumps, and 2.5 to 2.9 with high-pressure rotative engines.

Pressure-regulators for Pumps.—A steam-pump should be provided with a speed-governor and a pressure-regulator. The design is to place on the pump a diaphragm or a separate piston connected by a spindle with the valve supplying steam to the pump. When the water pressure becomes excessive it acts upon

a balanced diaphragm which opens the steam-pipe, and admits a greater supply to the pump.

A check-valve should be placed in the pipe between the pump and the delivery stand-pipe, to prevent the return of the water in the event of the pump stopping operations. A relief-valve on the water-pipes will also prevent any injury from an excessive pressure or sudden shock.

The Speed of the Steam-pump.—Pumps are usually started by a process called priming, which consists in removing the air from the barrel, filling the pump with water from a discharge-pipe until all the suction space is charged.

The speed may be varied, but is not advisable beyond 100 feet per minute as the standard rate. The water-pipe is calculated for a flow of 200 feet per minute. The piston speed is limited by that of the possible velocity of entry of the suction. A speed in excess of this not only results in water-ram, but reduces the capacity of the pump and increases the difficulty of a prompt seating of the valves. Any delay in the closing of the valves results in a slip of the water into the cylinder. The slip should not exceed 3 per cent of the piston displacement. A large number of small valves is preferable to a few large ones, so far as the amount of slip is concerned. The area of the suction-valves should be 40 per cent of the water-piston area. Their diameter should not be over 4 inches.

The Suction Height.—The size of the suction-pipe should be as large and as short as possible, that the pump may be able at high speeds to obtain as much water as it can deliver with a given speed of plunger. Its size will depend upon the length of the pipe between the pump and the water level, as well as the difference in elevation between the two. The former determines the frictional resistance, and the latter the maximum velocity of inlet entering the pump, due to the atmospheric pressure.

As the reciprocating piston cannot produce a vacuum lower than 3 or 4 lbs. absolute pressure, the atmosphere cannot lift the water higher than 28 feet in the suction length. To obtain a velocity of entry the pump must be placed nearer to the water

FIG. 146.—Compound Duplex Externally Packed Plunger-pump, with Ejector Condenser and Air-pump.

level than this suc.ion height. As it is desirable to have a high speed of transmission, the pump should be lowered to such a position as to give a velocity of at least 25 feet per second. Hence the maximum suction-lift of 14 feet should not be exceeded if high speed is desired.

The height to which the water is raised is equal to the level of the suction-lift below the pump plus the height to which it is forced above the pump. This height multiplied by 0.433 is equal to the head acting on each square inch of the water-piston. The area of the water-piston multiplied by this weight determines the total resistance. The steam pressure required in the pump per square inch of steam-piston is equal to the resistance just determined divided by the area of the steam-piston. This quantity multiplied by *m*, the efficiency of the pump, determines the actual steam pressure necessary to overcome the resistance of the column of water. Thus a 6-inch steam-piston and a 4-inch water-piston having areas of 28.27 and 12.57 sq. in. respectively, with the water pressure due to 100 feet of suction and force head, or 43.3 lbs. per square inch, the net steam pressure theoretically required will be 19.4 lbs. per square inch; allowing for a mechanical efficiency of .70, the steam pressure then must be 28 lbs. above that of exhaust.

Motor Fluids other than Steam.—Compressed air, oil, and water are used also as water agents in the driving of the direct-connected pumps. The oil-engine is a simple, self-contained, and portable motor of great value. An oil-engine of high economy is illustrated in Fig. 147.

Compressed air is used in the same manner as steam in the power cylinders, by utilizing its elastic property. A pump driven by air cannot be drowned out as is a steam-pump, and does not require as much attention as the latter. A high-class modern compressed-air plant will compare very favorably in commercial efficiency with underground steam-pumps. In one case 280 horse-power was consumed in raising 400 gallons per minute against a head of 120 lbs. per square inch, showing an efficiency of 9 per cent.

There are two other classes of air-pumps: the displacement pump, in which the water is displaced or expelled by entering

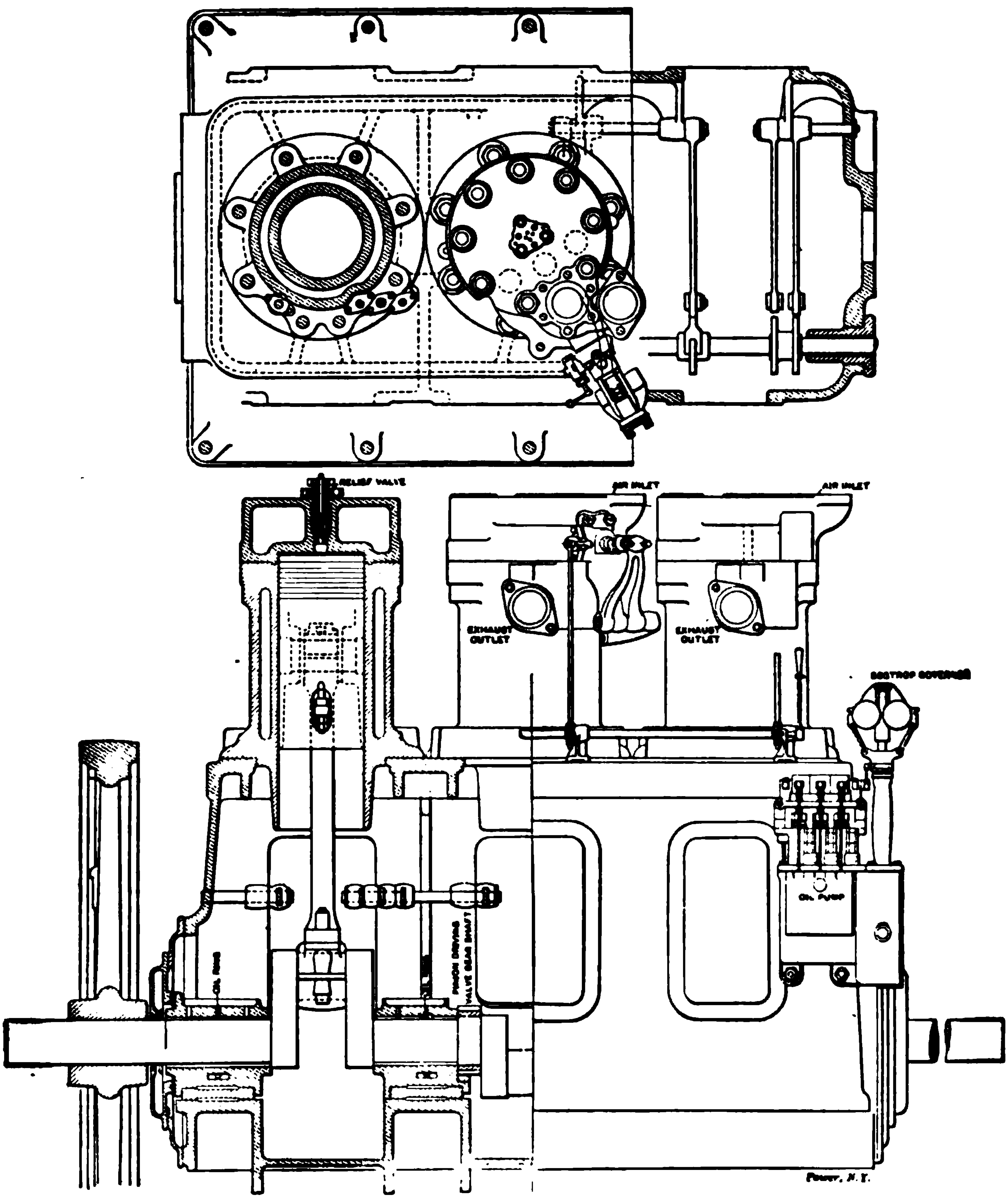

FIG. 147.—The Diesel Motor.

compressed air; and the air-lift pump, in which the water is raised by pressure and the expansive force of the air.

The Displacement Air-pump.—Two cylinders are provided side by side with an air-valve immediately above them. Compressed air is automatically admitted, first to one cylinder, then to the other, and the air correspondingly released. The pressure of the air on the surface of the water in the cylinder forces the latter through its discharge-valve to a height depending on the air pressure.

These displacement pumps, or tanks, may also be arranged in multiple series placed at various elevations between the bottom and the top of the shaft. At each station the displacement is effected in the manner indicated by a pressure sufficient to raise the water to the nearest tank above.

The Air-lift Pump.—In this extension of the displacement system air is delivered from a necessarily small pipe into the stand-pipe, where the water column is broken into short lengths by small volumes of entering air, whose expansive power overcomes gravitation. The efficiency of the system is regarded as low.

Water-pressure Engines are located at the mine below some source of supply. The water flows under pressure against the piston of the pump, which communicates the power to the rods of the Cornish system, or to a direct-connected plunger which lifts and forces the drainage from the mine (Fig. 148). If the engine be placed underground, a still greater head is obtained for power, the discharge being effected at a tunnel level intermediate between the shaft mouth and the sump level. This is the method at the Comstock mines. The great trouble with these engines is in the valve-gear, which is more complex and must be more nicely balanced for an incompressible fluid like water than is necessary for an elastic fluid like steam or air. The sudden shutting of the valves produces a not inconsiderable concussion in the inelastic fluid, which is entering the cylinder at a high velocity. Unless delicately manipulated the valves fail to operate.

They will operate at any reasonable rate within the limits of the valve movement. The fluid, being inelastic, requires a very heavy construction to resist the static pressure and the water-

FIG. 148.—Hydraulic Pumping-engine.

ram. Ample time must be allowed for pause at the end of each stroke. The hydraulic engines are expensive plants. They cannot be used for shaft-sinking, as the working pressure cannot be altered to meet the increasing resistance. Their field as prime motors at the surface has been reduced by the much cheaper and more durable high-pressure impulse-wheels, driving centrifugals.

Formulæ for Pump Calculations.

Let c = diameter of the water-cylinder, inches;
N = number of strokes per minute;
k = diameter of the steam-cylinder, inches;
s = stroke, inches;
d = diameter of the pipe, inches;
h = height of head lost due to friction in the pipes, feet;
L = height of the lift including the suction, feet;
G = discharge in gallons per minute;
Q = cubic feet per minute;
f = pump capacity less the slip, in per cent.

Then

$$G = 0.0034c^2sNf; \quad Q = 0.000454c^2sNf.$$

The pressure per square inch of water-piston $= 0.434L$.

The pressure per square inch of water-piston on the bottom of the pipe $= 0.341Ld^2$.

Total pressure on the water-piston $= 0.341Lc^2$.

$k^2p = 0.341(L+h)c^2$.

The work of the pump $= 0.000253GL \div f$.

The I.H.P $= 0.000253G(L+h) \div mf$.

EXAMPLES.—1. Let it be required that 100 gallons of water weighing 833 lbs. be raised 200 feet in one minute. The number of ft.-lbs. of work necessary will be 166,600 ft.-lbs. When the mechanical efficiency is 0.6 the I.H.P. = 8.43.

2. Required the horse-power necessary to furnish 50 cubic feet of water through 400 feet of 1½-inch pipe and a suction-lift of 4 feet.

According to the formula, Chapter VI, the head of friction is equal to 5 lbs. per square inch; the total pressure due to the column of water is 173.2 lbs. per square inch; the velocity through a 1½-inch pipe is equal to 80.3 feet per minute; whence the work done is 178.2×80.3 feet = 14,000 ft.-lbs. The steam end must be capable of furnishing more power than this, and if the

initial steam pressure be 100 lbs. absolute, with a back pressure of 18 lbs., the dimensions of the cylinder with $m=0.50$ will be 6×12, assuming a maximum piston speed of 100 feet per minute.

The pump receives its supply from a boiler which is independent of those feeding the hoisting-engines, because the intermittent work of the latter causes such changes in steam pressure as to seriously affect the speed of the pump.

3. A mine delivers 1900 gallons per minute. The depth of the shaft is 468 feet. Required the size of the pump-cylinders under a boiler pressure of 100 lbs. (gauge) and a $\frac{1}{3}$ cut-off, back pressure being 16 inches of mercury. Efficiency, 60 per cent.

As the ordinary piston speed is 200 feet per minute, the discharge of 4.227 cubic feet per second may be delivered at the same speed in the pipe, which is then of a diameter of 15 inches; or if a 10-inch discharge-pipe be employed, the velocity therein would be 464 feet. The loss of head would be respectively 1.56 and 11.86 feet. Total head being then 470 or 480 feet, the work of raising the 1900 gallons would be 7,441,500 and 7,599,840 ft.-lbs. respectively. The two steam ends must be capable of 12,402,516 and 12,666,400 ft.-lbs. From the table in Chapter IV, the mean pressure corresponding to a cut-off of $\frac{1}{3}$ is 0.726 for 1 lb., and 83.27 lbs. for 114.7 absolute.

Let the average effective pressure be 64 lbs., then the diameter of each steam end should be $12\frac{1}{2}$ inches. Assuming a stroke of 24 inches, each stroke represents 2.53 cubic feet, and the diameter of each water-cylinder would be $10\frac{3}{4}$ inches. If the minimum effective pressures upon the two steam-pistons be taken (25 lbs. per square inch on one and 64 lbs. on the other), the diameter of the water-cylinders should not exceed $7\frac{1}{2}$ inches. This discussion neglects the inertia of reciprocating parts.

If the discharge-pipe be assumed at 6 inches in diameter, it would entail a loss of head of 151.47 feet, requiring 475 horse-power.

The ratio between the diameters of the two ends of a steam-pump is about 1 : 2 for the smaller sizes, and the steam end is three times that of the water end in the large sizes.

4. What volume could be raised by a double-acting steam-pump having water-cylinders 8 inches in diameter, and the steam-cylinders 18 inches, with a 2-foot stroke? Piston speed 200 feet per minute. 139.4 cubic feet.

5. What should be the effective steam pressure to discharge the water, assuming an efficiency of 50 per cent and a shaft 400 feet deep? Assuming a discharge-pipe of 5 inches diameter, p is 83.7 lbs. per square inch, the loss of head being 88.22 feet.

Power-driven Pumps.—Where it is not possible or desirable to connect the water-plungers directly with an engine, power may be communicated to a pulley on a rotary shaft by a belt

or wire rope, or to an electric motor. If the distance between the engine and the pump is very great, electricity is the agent for the motor. If the distance is slight, a belt is employed, with or without gearing, to reduce the speed of the motor to that of the pump, which is always low. Power-pumps are built with more than one trunk-cylinder whose pistons are single-acting, the shaft being a multiple crank. With three plungers the cranks are 60° apart, with four they are 90°, etc. Trip-levers can be attached to relief-valves, so that, when the pressure becomes excessive, they will permit the water to escape and simultaneously close the suction-valve.

The cylinders are generally vertical, though in the Riedler pump they are horizontal. Double-acting duplex or triplex pumps can also be had, of the power-driven type, which are used for pressures up to 700 lbs. in the anthracite region.

The power-pump is of practical utility and simple, occupying less floor space for a given capacity than the direct-acting. It is of low speed and is therefore geared down from the driving source. Its loss by friction is a little greater than in the direct-connected pump. Where fuel is expensive it has a distinct advantage in the point of steam economy, for the direct-acting type cannot use steam expansively. The power-driven pump may be operated by a modern automatic cut-off engine and develop power with less steam than the direct-acting. The latter cannot use the steam expansively, as is possible with the engine and a fly-wheel-crank-driven pump.

The Riedler Pump has mechanically operated valves which will open and close at high rotative speed. Its capacity is large for the space it occupies and it requires little foundation. At its maximum speed of 300 r.p.m. it can raise water 1000 feet. The single-acting plungers are installed two or three on a line, to lighten the work and economize power.

Electrically Driven Pumps are necessarily of the geared-crank type, for this power is not adapted to reciprocation. Either continuous- or alternating-current motors are suited for them. The main requirements are a practically constant moment or

torque, and a nearly constant speed. With a continuous current the compound-wound motor is employed; if a variable speed is demanded, a rheostat in the shunt-field will suffice. Its advantage over an ordinary shunt-motor is that its series-coils obviate the wide variation of current which would occur with the latter when passing through the different points of the pump cycle. It will not race if the pump happens to lose its water. Preferably, it should not be enclosed. In the shunt-wound motor the field resistance will rise as the enclosed machine gets hot, thus causing higher speed and armature current. The difficulty can be avoided by a few additional turns of series winding.

FIG. 149.—Electric Power-pump.

With the alternating current the short-circuited squirrel-cage type of armature induction-motor, designed for any desired turning moment and operated at a constant speed, is well adapted to drive the pumps.

The leakage loss of electric pumps is small and their friction loss low, while the expense of maintenance and repairs is a

minimum. One compartment may be saved in the shaft, since the wires and the discharge-pipes can be placed in either of the hoistways without interference. In common with the other motor-driven pumps, it is portable. It is started and stopped easily by the ordinary switch with an automatic release-switch.

Like the power end of the air- and water-driven pumps, the motor may be operated even under water, for it can be completely housed with its sinking-pump.

Electrically driven pumps require automatic switches which, when the water level reaches a given point, engage a lift that moves the switch and opens or closes the circuit, as may be desired.

Rotary Pumps are employed for lifting and forcing water against low head. This type of pump is compact and self-contained and will deliver more water for a given weight and space occupied than will reciprocating pumps. It is usually driven by belting or wheel gearing. Rotary pumps may be divided into several classes, according to the forms of the pistons or impellers, or according to the arrangement of the butments. Two rotary impellers receive, in the space between them from the suction below, a volume of water which is carried with the evolution of the impellers around to the upper side of the pump. The butments receive the force of the water, and prevent the latter from being carried around the cylinder, thus compelling it to enter the delivery, where it is discharged. The Sturtevant blower has two rotary pieces, and the Root three lobes.

In some pumps the butments are movable and are arranged to be pushed back, as they revolve, to allow the piston to pass a given point; in others the pistons give way when passing fixed butments; and in others the pistons are fitted with a movable wing, as in the fan, which slides radially in and out when passing the butments. These pumps have no packing or springs, are quite durable, but have a tendency to become noisy as the gearing wears.

The Centrifugal Pump.—This is the cheapest and, under certain conditions, the most efficient of the rotary variety of

pumps where the height of lift is moderate and large volumes are to be moved. Like all of the rotary type, it handles sandy or muddy waters with facility, and has a very short suction and only a moderate height to which it can raise water. It is a fan with a number of blades attached to a shaft, and is turned by an electric motor or steam-turbine at 500 to 800 r.p.m. The inlet

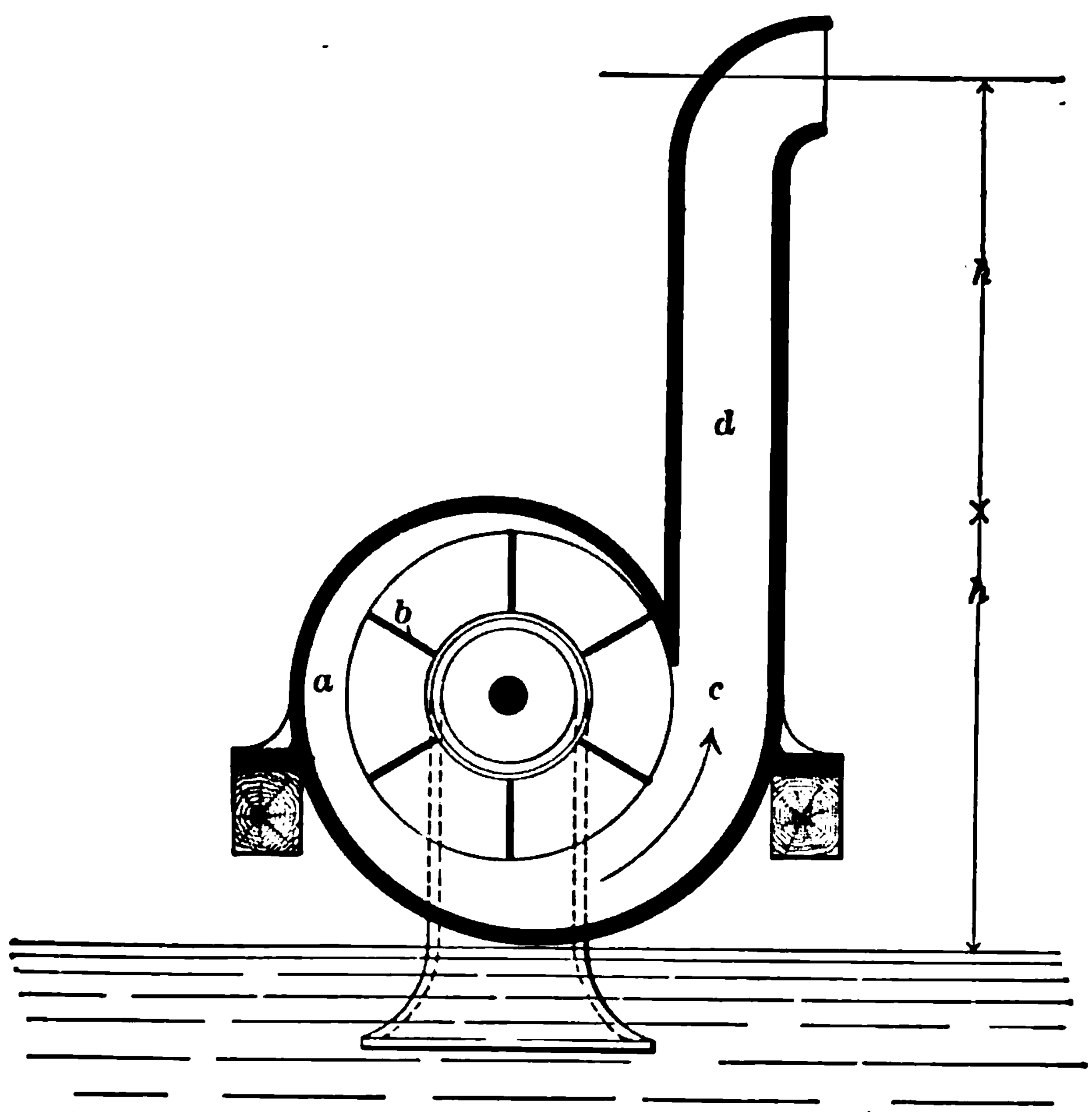

FIG. 150.—Section of a Centrifugal Pump.

may be on one side only, or on both (Fig. 151). With the latter the inlets and blades may be kept down to smaller sizes for an equal capacity. The water enters at the centre and is delivered at the circumference by the high speed of the blades. The blades are convex, radial, or concave. The former require a higher rate of velocity for a given lift than do radial blades. Their inner ends are curved forward slightly to scoop up the water with a minimum of shock. The inlet velocity should not exceed 3 feet per second. The pump is primed and started

by some form of steam-injector which expels the air and fills it with water.

The Theory of the Centrifugal Pump.—The water is driven through the fan partly by the pressure of its blades and partly by centrifugal force, and the water will escape with that force at *c* (Fig. 150), but with little rotary motion, and will rise to a height, h, corresponding to the centrifugal force. If there be no unavoidable efficiency losses, the relation between the total lift, h, and the peripheral velocity, V, is expressed by

$$h = 0.015625 V^2.$$

If the outlet for the water be of proper size and flared to reduce the velocity of discharge gradually, the energy imparted to it by the blades is converted into pressure head. With that due to centrifugal force, the head ideally obtainable from a periph-

FIG. 151.—Double-inlet Centrifugal Pump.

eral velocity, V, is $2h$ (Fig. 150). The usually imperfect pump, however, shows a relation between velocity and head of

$$V = \sqrt{64.4h + u^2},$$

in which u is the velocity in the discharge-pipe. The head, h, is only about 60 per cent of that theoretically due.

To increase the discharge for a given head means loss of efficiency, because the speed of the pump and that of the water issuing from the pipe are increased unnecessarily; a reduction of velocity is also attended with loss by fluid friction. Where efficiency is desired the design must be just suited to the precise conditions under which it is to operate. The first cost is low and its efficiency increases to 70 per cent at about 40 feet of lift, beyond which height the efficiency again diminishes, until at a height of 100 feet the efficiency is but 40 per cent.

Let N = r.p.m. of the wheel;
H = height of delivery in feet;
D = diameter of wheel in feet;
K = constant; 153 for small pumps and 187 for large pumps;

then $$N = K\frac{\sqrt{H}}{D}.$$

EXAMPLE.—A centrifugal pump discharges 180 cubic feet per minute at a peripheral velocity of 30 feet per second. Required the horse-power to drive the pump. The peripheral velocity of the water is 25 feet per second. Then momentum lost by the wheel per second $= \frac{3 \times 62.5}{32.2} \times 25 = 145.5$ lbs.

The work done $= 145.5 \times 30 = 4365$ ft.-lbs. per second $= 8$ horse-power.

The efficiency being assumed at 70 per cent, the horse-power applied to the pump is 11.5.

$$4365 \div 3 \times 62.5 = 23.3 \text{ ft.-lbs. energy.}$$

This would raise a pound of water 23.3 feet.

The Compound Centrifugal Pump.—These pumps are essentially of a high-speed type with low head of lift. A high lift is attained only by increasing the rotary speed. If, however, two or three wheels, each in its own chamber, are mounted in series on the same shaft, the lift of each single one is multiplied, while yet keeping the speed within moderate bounds (Fig. 152).

In such a multi-stage pump the water from the discharged chamber of the first impeller is led back to the suction-point of the next impeller through channels in the pump-casing. This is

repeated as often as there are impellers. Each pump of the series raises the pressure head from that of its suction. A three-stage pump, with each member capable of 100 feet of head, will then deliver water to a height of 300 feet with, of course, a corresponding increase of motor-power. These are much used in placer-mining service instead of a long pipe-line from a distant lofty reservoir.

The Efficiency of the Centrifugal Pump.—The efficiency of these pumps drops rapidly when the difference between the outlet and the inlet pressure heads exceeds 60 feet, and the efficiency of a multi-stage pump is about the same (0.85) as that of a single

Fig. 152.—A Three-stage Pump.

pump of the same construction, but the advantage of the former lies in the reduced velocity for the given height of lift. On the same work and within reasonable limits, the multi-stage centrifugal is slightly more efficient than the single pump, due to the decrease in frictional loss attaining the reduced rotary speed. The steam-turbine, or the electric motor, is capable of direct connection with the centrifugal pump, and. this combination

compares very favorably in steam consumption with the direct-acting type.

HORSE-POWERS AND FUEL REQUIRED FOR TWO- AND THREE-STAGE CENTRIFUGAL PUMPS FOR HYDRAULIC MINING OR FOR PUMPING PURPOSES.

Capacity.			Horse-power at 70 Per Cent.	Diameter.	Pounds Coal, 12 Hours.	Short Tons.	Barrels Oil.	Cords Wood.
Miner's Inches.	Gallons per Minute.	Cubic Feet per Second.						
50	562	1.2	41	5″	2,460	1.2	4.4	2
75	843	1.8	62	6″	3,690	1.8	6.7	3
100	1125	2.5	82	6″	4,920	2.4	8.9	4
125	1406	3.1	103	7″	6,150	3.1	11.1	5
150	1687	3.7	124	8″	7,380	3.7	13.3	6
175	1968	4.4	145	9″	8,610	4.3	15.5	7
200	2250	5.0	164	10″	9,840	4.9	17.8	8
225	2531	5.6	185	10″	11,070	5.5	20.0	9
250	2812	6.2	206	10″	12,300	6.2	22.2	10
275	3093	6.8	227	12″	13,500	6.8	24.4	11
300	3375	7.4	246	12″	14,760	7.4	26.6	12
350	3937	8.6	267	12″	17,220	8.6	28.9	14
400	4500	9.9	328	14″	19,680	9.8	35.5	16
500	5625	12.4	410	14″	24,600	12.3	44.4	18

A single-stage pump making 1500 revolutions per minute, with 55 horse-power steam-turbine, delivered 1700 gallons per minute with a lift of 100 feet. The diameter of the plunger is 13¾ inches. A two-stage centrifugal pump, 9 inches in diameter, was making 2000 revolutions per minute with a Dela valve steam-turbine running at 20,000 r.p.m., and raised 250 gallons per minute 700 feet. The tests of these showed the efficiency of the wheel and turbine to be 75 per cent in the first case between the limits of 1400 and 1800 gallons per minute. The duty of the pump per thousand pounds of steam operated with a condenser was nearly 62,000,000 ft.-lbs. The two-stage pump showed a duty at 250 gallons per minute of 48,800,000 ft.-lbs. per thousand pounds of steam. Forty-two pounds of steam were used per water horse-power.

A test of a single-stage pump direct-connected electric motor of 20 horse-power delivering 1200 gallons per minute at 2000

revolutions for a lift of 45 feet showed an efficiency of 76 per cent. The diameter of its wheel was 8.3 inches.

EXAMPLES.—1. What will be the size of the discharge-pipe, d, the wheel and its rate of revolution, to deliver 1000 gallons per minute for a height of 50 feet?

$$d = 0.25\sqrt{-G} = 0.25 \times 31.6 = 7.8 \text{ inches pipe;}$$
$$\text{wheel diameter} = 2 \times 7.8 = 15.6 \text{ inches;}$$

and

$$N = 187 \times \frac{\sqrt{50}}{1.5} = 880.67 \text{ r.p.m.}$$

2. Required the size of a three-throw electric pump, the horse-power and current at 550 volts for a motor of 90 per cent efficiency, to deliver 240 gallons 300 feet high. Speed of piston 60 feet per minute. Let $f = 0.97$. Then $c = 5.86$ inches and the stroke $1\frac{1}{2}c = 8$ inches.

The work of pumping is $240 \times 8\frac{1}{3} \times 300 = 590,000$ ft.-lbs. Assuming the inertia, friction, etc., to be 50 per cent, i. requires 36 H.P. at the motor-shaft. With 90 per cent efficiency, this $= 29,840$ watts, which at 550 volts $= 54.3$ amperes.

3. It is desired to raise 90 gallons of water per minute up an incline of 1 in .5 which is 1200 feet long. Assuming a discharge-velocity of 3 feet per second, what will be the size of the pipes and of the pumps and motor? If the three-throw pump is employed, each pump raises 30 gallons; with 20 per cent of slip, 36 gallons must be raised about 60 feet per minute, with a wheel-diameter of 4.25″. At a stroke of 8 inches and 90 revolutions per minute, the diameter o the discharge-pipe will be 3.875 inches to deliver 90×0.16 cubic feet per minute. A pipe of 4 inches diameter will lose, according to the formula in Chapter VI, 1.22 feet of head for each 100 feet of length at the assumed velocity. The loss of head due to friction is then 14.6 feet. The work of lifting the water, including the above loss, is 190,980 ft.-lbs. per minute and the horse-power is nearly 6. Allowing a friction of gear and starting of 50 per cent, 12 B.H.P are required. The motor will run at 630 revolutions per minute with a gear of 7 to 1. If the latter has an efficiency of 80 per cent and receives a current at 380 volts, 24.5 amperes will be required to deliver the 10 B.H.P. at 380 volts. The wire for this current must be 19/19 and the drop in voltage for the total length of wire down the slope will be 24.5×1.786, which equals 44 volts per mile and 20 volts for 2400 feet. The voltage at the bottom is therefore 400.

The Pulsometer.—This useful apparatus raises sandy or acid waters to limited heights where economy of fuel is less im-

portant than quickness of installation. It is perfectly free from risks of breakdowns, and is employed in open works. *AA* (Fig. 153) are two chambers into which steam enters according to the position of the ball, *C*, which oscillates from one side to another of their necks. Through the inlet passage, *D*, the water enters into *A* by opening the valves, *E*. *H* is a delivery passage communicating with each chamber through openings and valves, *G*.

FIG. 153.—The Pulsometer.

Steam enters at the top, passes into a chamber uncovered by the ball-valve, and presses upon the surface of the water, forcing it down and out through the discharge-valves into *H*. When the water-line falls below the discharge outlet, the steam above condenses, a partial vacuum is formed, and its pressure suddenly falls again. Meanwhile, with the collapse of the steam, the ball-valve is thrown to close that chamber, and admits steam to the other side. Here the water is expelled, in a similar manner, out through *H*, while at the same time water has entered through *D* to be expelled from the other side.

The limit of lift is about 30 feet, and the capacity of some pulsometers is often 1000 gallons per minute.

Siphons.—Though not a water-raising appliance in the proper sense of the term, since water by this apparatus can be lifted or transported over an eminence not exceeding 28 feet in height and discharged on the other side of it at a level lower than that of the supply, a siphon can find application for forming a water communication over a slight elevation between two distant points. This height will be further reduced by an amount necessary to cause the required velocity of flow and to overcome the frictional resistance. Formulæ in Chapter VI will determine the losses due to friction in the given case, *L* and *d* being given.

When the siphon is not too long, and when the acceleration head is sufficient to give the water a considerable velocity, the

air, entrained by the rapid current, may be carried out at the end of the discharge branch, if the latter is not too steep. In most cases, however, it is necessary to provide artificial means to remove the accumulated gases, either periodically or continuously. A hand-pump is usually employed for this purpose, its suction being connected to the highest point of the siphon, and operated as occasion requires.

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CHAPTER XII.

MINE-GASES.

Ventilation as an Economic Proposition.—The ventilation of a mine is a matter of very great concern, not only from a humanitarian standpoint, but from an economic point of view. No shaft or tunnel can be carried more than 200 feet beyond an opening without some special means of stirring and freshening the stagnant air. The men should not be compelled to work in the hot atmosphere of a stove or room vitiated by the variety of gases given off from and by coal, powder, lamps, respiration, rotting timbers, and decomposing ore. These gases cannot support combustion, nor can they be inhaled with impunity; and such an atmosphere is unfit for respiration, being deficient in oxygen, as well as by reason of the presence of these gases. In coal-mines additional peril accompanies some of these gases, which with the air form explosive mixtures, which, bursting into flame, destroy everything in their path and emit dense volumes of poisonous fumes that are fatal to all who have escaped the shock.

Recognizing the pecuniary value of the life and energy of a miner, the statutes are becoming more and more rigorous in the insistence of safety and hygienic measures. Methods are specified for rendering the noxious gases harmless, and officials are given sufficient authority to suggest needed improvements and to punish non-compliance. Not only are the miners benefited by the diminishment of risk, but also the operators, who profit in the increased energy of men working under favorable circumstances. The illumination is better, smoke clears away quicker, and the men are capable of a full day's work.

Comparison of Metal- and Coal-mines.—There is a marked contrast between the requirements of vein-mines and those of coal or other mines in flat beds, which latter usually have two shafts connected by a labyrinth of workings on nearly one level. The ventilation of metal-mines presents by no means the same difficulty as that of gaseous collieries, since as a rule the former class of mines requires but a small supply of fresh air, abundant enough for the health, comfort, and effective work of the men and for the removal of the dead air vitiated by various causes. Trust is placed in natural means of circulating the air by the winze communication of the different levels. This is certainly inadequate, and the lower mortality in well-ventilated coal-mines is doubtless due to their better ventilation. In coal-mines large volumes of fresh air, additional to that required for the men, are necessary to carry off the fire-damp, choke-damp, and other gases, both noxious and inflammable, though no serious results may occur from their presence in large quantities. These gases are continually being evolved from the coal and constitute an ever-present danger to coal operations, but do not threaten metal-miners. Nevertheless, the latter should remember that the inflammable gases of coal-mines, from which they are comparatively free, are not the only ones to be guarded against. The cloud of dust formed by blasting the mineral, and the carbonic acid gas, are dangers equally insidious to the health of those inhaling it, which cause as many deaths as do the small explosions which occur in coal-mines.

Gases in Coal-mines.—In addition to the gases found in a normal atmosphere, there exist in mines several gases which are always the result of decomposition and combustion. Pursuing the nomenclature of early mining, they are designated as "damps." For example, carbonic acid, CO_2, is known as choke-damp; carbonic oxide, CO, as white damp; sulphuretted hydrogen, H_2S, as stink-damp; carburetted hydrogen or marsh-gas, CH_4, as fire-damp or fulminating damp; air vitiated by breathing, having therefore a deficiency of oxygen, as black damp; and the residual gases of an explosion, after-damp. In

metal-mines carbonic acid and sulphuretted hydrogen are the only gases met with. Both are heavier than air and naturally will be found near the floor of the workings. Coal-mines, however, are troubled with additions and emanations, sudden or continual, of carbonic oxide and marsh-gas, which two, being lighter than air, are to be looked for in the upper portions of the workings, near the roof.

Composition of the Atmosphere.—Every hundred grains of the atmosphere contain 76.84 grains of nitrogen, 23.10 grains of oxygen, and 0.06 grain of carbonic acid, occupying respectively 79.02 per cent, 20.94 per cent, and 0.04 per cent of the original volume. These gases are not in chemical combination, but as a mechanical mixture from which the oxygen may be extracted under given conditions.

Oxygen, which is colorless, odorless, and transparent, unites with all other chemical elements, forming various chemical compounds. All the ordinary phenomena of light, heat, and fire are the result of the union of other elements with oxygen. This active principle is essential to life and to all other processes of combustion, and when combined with carbon forms choke-damp or white damp; with pyrites, a common mineral in coal, sulphuretted hydrogen; and with marsh-gas, water or steam and carbonic acid. An atmosphere deficient in oxygen will no longer support combustion, and a lamp or flame immersed in it is extinguished; so, too, air-breathing animals are suffocated when the percentage of oxygen in the air is below 15.

Nitrogen, the predominant constituent of unpolluted air, is colorless, odorless, and incapable of supporting combustion or animal life. It is inert in its effect upon the system, but extinguishes flame immersed in it.

Carbonic Acid, which is not an important constituent of the atmosphere, is, however, to be expected in all abandoned and unventilated places. It is given off in quantity by the respiration of men and animals, by the combustion of lamps, fuel, explosives, timber, and organic matter, and by all substances in a state of decay or fermentation. It, also, is colorless and odor-

less, and at a normal pressure of 30 inches of mercury and a temperature of 32° F. weighs 128.45 lbs. per thousand cubic feet, as against 81 lbs. for an equal volume of air under the same conditions. It is easily detected, for, being incapable of sustaining combustion, light burns dimly when immersed in it. When, therefore, the gas is revealed by this action on a flame, the atmosphere is at least unfit for respiration. The gas has an injurious effect upon the human system. Air containing 2 per cent of it produces an overwhelming depression upon those breathing it; with 6 per cent, lights are extinguished; and with 10 per cent, it is positively fatal by producing suffocation. As it accumulates in the lower part of idle workings and where the air is stagnant, no one should venture into abandoned works without having previously tested by a flame the condition of their atmosphere, and if the light is extinguished, the poison must be swept out by a strong current of air, which, in shafts, may be incited by rapidly raising and lowering a bundle of hay, which is then ignited. Small volumes of carbonic acid may be removed from the atmosphere by the use of absorbents like lime or ammonia. Its presence may be chemically detected by the milkiness produced in a test-tube containing baryta hydrate.

Sulphuretted Hydrogen, which at 30 inches barometric pressure and a temperature of 32° F. weighs 94.62 lbs. per thousand cubic feet, is an extremely poisonous and common gas occurring in mines. As it is the result of the decomposition of pyrites, always accompanied by heat, this gas is a warning of incipient fires. In the gob and abandoned portions of the mine the pyrites and coal-waste subjected to pressure from the roof, and in the presence of moisture, decompose with the development of heat and possibly flame, of which the presence of sulphuretted hydrogen is an important indication. Being colorless but strongly odorous, its presence is readily detected. It does not support combustion, but is itself inflammable. A flame will burn in a mixture of it with air, but is extinguished in an undiluted atmosphere of this noxious gas, which in this state is fatal to life.

Carbonic Oxide, which is claimed to be a normal gas exist-

ent in coal-mines, weighs 78.305 lbs. per one thousand cubic feet at a temperature of 32° F. and a barometric pressure of 30 inches. This gas has neither color, taste, nor smell, and is exceedingly poisonous. One half per cent of this gas in air renders the atmosphere fatal to life if breathed for ten minutes. It acts upon the system by combining with the oxygen absorbed in the blood, forming a stable compound reducing the hæmoglobin, and insidiously and surely destroying the blood and tissues. As carbonic oxide is an unstable compound usually resulting from the imperfect or secondary combustion of a gas or of carbon in the deficiency of oxgyen, the existence of this gas in mines is doubted. It is only barely possible that it might be found in the goaf, where the oxidation of pyrites and the absorption of oxygen by the fine coal-dust may have depleted the air of its vital element, thereby giving rise to a sufficiently high temperature to incite combustion with the development of carbonic oxide.

Light Carburetted Hydrogen or Marsh-gas is a stable, never-failing constituent among the products of dry distillation of organic matter, and exists as the predominant constituent of the compound gas known under the general term of fire-damp because of its ready ignition by flame with a mixture of air. Its weight at the normal barometric pressure and a temperature of 32° F. is about 45.22 lbs. per thousand cubic feet. It is absorbed in the coal, diffused through its pores, collected in crevices or cavities, and even stored up in reservoirs, having been exuded from the coal in the early stages of decomposition of the organic source of the coal, or expressed during the geological movements of the earth's crust. It is not a constituent of the coal, but is entirely free from chemical combination with it, and continually exudes with greater or less violence; it is liberated in volumes by falls in the mine-roof, by squeezes or creeps, and by any sudden fall in the barometric pressure. The deep portions of coal-seams appear to be more heavily charged with this compound gas than are workings in coal-beds so close to the surface as to allow of its escape. No coal-seam should be regarded as free from liability to irruptions of this gas.

The Occlusion of Gas in Coal.—Marsh-gas exists free in the pores of all coals, but must not be confounded with the gases chemically compounded with the coal. It is under high pressure, and is given off into the workings of the mine with greater or less violence from the fissures or crevices, sometimes without warning, sometimes accompanied by the heaving of the floor and the trembling of the roof, but always more or less distinctly audible. The volume so emitted from the coal varies in amount from 9 to 100 cu. ft. per ton. From anthracite the discharge is most copious, and from bituminous coal the least.

Frequently it escapes into the mine without warning; its presence is not always detected or manifested, as it accumulates in a nook or under a platform until some untoward circumstance brings it into contact with a naked light. The slight hiss accompanying its exudation is hardly enough to be distinguishable. These "blowers," of all sizes, up to the outbursts that for a time overpower the ordinary ventilating current, contain 90 per cent of marsh-gas, and may be liberated anywhere and at any time. For a long time it has been recognized as a constituent of the gases entering mines. It escapes at various springs and salt-mines; it has fed the sacred fires of Baku and the mud-volcanoes of Bulganak; it has been found in the Silver Islet mine, in the iron-mines of Alsace, and in the lead-mines of Tuscany; and some years ago an explosive gas was met with in driving the lake tunnel at Chicago. It constitutes from 40 to 90 per cent of the natural gas, and is obtained among the volatile and combustible constituents in the ultimate analysis of coal. While sinking shafts through porous strata fire-damp has been encountered, and precautions are therefore necessary in regions of natural gas, or in formations immediately above the coal horizons.

Outbursts of Gas from the Coal.—Measurements made by the Royal Commission on Accidents in Mines have revealed the pressure of gas to be frequently as great as 450 lbs. per square inch. Sudden outbursts may therefore be expected when escape is afforded, and these, according to the treatise of M. Roberti Linterman, preponderate in coal-seams disturbed by

faults, foldings, or thinnings-out, and are influenced by the dip of the seam. They occur without any premonitory symptoms and even in districts heretofore free from gas. They are most voluminous in the periods of pillar-robbing or during the exploration of virgin ground. The disengagement of gases increases both in intensity and frequency with the depth of the workings and with the presence of permeable masses surrounded by rock-masses so hard and compact as to constitute effective barriers against fire-damp. In every district are found one or more infested zones of gas.

Owing, therefore, to the inevitable occurrence of this inflammable gas in all coal-seams, and the uncertain quantity which may be thrust into contact with the flame of the illuminating-lamp or of the explosive employed in the mine, fire-damp is the dread enemy of coal-miners. The amount of gas which renders the mine unsafe cannot be stated, for, while an atmosphere containing 2 per cent or more is neither injurious to life nor dangerous in mines, nevertheless its presence to that extent in the air discharged from the mine into our atmosphere indicates a probably excessive accumulation in some of the workings. The Coal Commission of Austria, in its classification of mines according to the composition of the air at the outlet of their ventilating-shaft, regards those having more than 2 per cent of fire-damp and carbonic acid in their return air as fiery, and those having less than 1 per cent of gas admixture as safe.

The Effects of Mine-gases upon Life or Flame.—Mine-gases mixed with air do not have the same effect upon life or an illuminating-flame. In an undiluted state each and all extinguish flame and do not sustain life. When mixed with air the carbonic oxide and sulphuretted hydrogen are poisonous, while at the same time they support combustion. The marsh-gas is inert in its influence upon life, but is capable of ignition; while the carbonic acid is depressive in its influence both upon flame and upon life. There are hence two methods of discovering the probable unsuitability of air for respiration—the extinction of flame by an excess of carbonic acid, and the flame aureole from the com-

bustion of carbonic oxide, the odor of sulphuretted hydrogen being a sufficiently strong warning without other index of its noxious presence. Fortunately, however, the presence of carbonic oxide need only be feared after a fire-damp explosion or after a blast of one of the lower grades of black powder.

Explosive Gaseous Mixtures.—Marsh-gas with air will burn freely, but when the proportion of gas reaches a certain specific amount the ignition may take place rapidly, and if the products of combustion cannot escape equally rapidly, explosion ensues, the force of the explosion and the dangerous degree of dilution of gas varying with the different gases.

The range of proportion of gas dilution between the lower and the upper explosive mixture is least in the case of fire-damp, which may vary in amount between 5 and 13 per cent of the total volume of air; is greatest in the case of carbonic oxide, which will explode with any mixture containing between 13 and 75 per cent; while with sulphuretted hydrogen the variation lies between 9 and 28 per cent of gas in the mixture. The gases are mentioned also in the order of the decreasing danger of explosibility, the first offering the greatest risk. When the atmosphere in which a flame is immersed contains a percentage of gas approaching the explosive limit, the cap and nimbus become large, and the flame almost invisible. The rapidity with which the ignition is propagated depends upon the nearness of the proportions of the admixture to the figures given, and when the maximum explosive ratio as indicated above is reached, the propagation is instantaneous, and the concussiv force of the explosion is also a maximum; as the percentage of gas recedes from these ratios or increases beyond the limiting explosive proportions, the violence of the explosion decreases. When either of the gases is undiluted with air, the light placed in contact with it is extinguished. In other words, a flame may be immersed in the workings filled with fire-damp; and if the line of demarcation between the air and the gases is "sharp," no explosion will ensue, but the flame will enlarge and, after a little fluttering, become extinguished. This is equally true of the

other gases, though the percentage of their accumulation in mine-workings is so low and their diffusion so perfect that no undiluted accumulation of them is likely to ensue. When ignition or explosion takes place, the products of combustion are termed the "after-damp."

But one means is available for the prevention of excessive accumulations of explosive gases, and this consists in supplying a copious volume of air and a thorough system of distribution which will dilute the noxious emanations below the danger-line. This will require careful examination daily of all suspected places and the enforcement of rigid discipline. The airways must be ample in area to allow the requisite volume of air to pass without producing a current of high velocity. In mines using the common Davy or the Clanny lamps, the maximum velocity admissible is 300 feet per minute. Where bonneted lamps are in use the velocity may be 800 feet without fear from explosion. Immunity from explosions is possible only by adherence to these requirements and care in the use of blasting agents.

Black Damp; After-damp.—This residual gas, after an explosion or ignition of gas, will extinguish a flame because of the deficiency of oxygen, and is theoretically composed of 52 per cent of nitrogen and 48 per cent of carbonic acid. The percentage of mixtures of oxygen, nitrogen, and carbonic acid in an atmosphere extinctive of flame is almost identical with that of the air expired from the lungs of men. Dr. F. Clowes, as the result of a series of experiments in relation to the lighting of mines and the behavior of lamps, has ascertained that the percentage composition of the residual atmosphere in which flame was extinguished, is, oxygen 15.7, nitrogen 81.1, carbonic acid 3.2, while that of the average exhaled air is oxygen 16.15, nitrogen 79.9, and carbonic acid 3.95.

Respiration in such an atmosphere is difficult, and produces unconsciousness, followed by marked panting in the effort to supply oxygen to the lungs. The countenance becomes swollen and livid, the features distorted, the eyes protrude, and as

asphyxia is pronounced, there is a sudden cessation of the pulsations of the heart and of respiration.

Usually some steam and carbonic oxide exist in the after-damp in proportions varying with the temperature of explosion and the initial proportions of air and explosive gas. In the presence of the latter, even in minute quantity, is the gravest danger. A reduction of oxygen to 14 per cent or below also causes disastrous results.

To the presence of this carbonic acid is attributed the loss of many lives in a mine explosion—more, in fact, than are the result of its concussion or of contact with its flame. The perfectly natural appearance of the body, lying often by a lamp still burning, proves the cause of death to be some insidious poison which is combustible, not asphyxiation or concussion. All members of rescuing parties entering the workings thereafter should take due precautions against the inhalation of this carbonic oxide gas.

Treatment for Asphyxiation.—Persons overcome by any gas may be revived by blowing oxygen into one of the nostrils, the other being closed, and by inducing artificial breathing. Epsom salts, and water acidulated with vinegar, are better than alcoholic stimulants. The warmth of the body should be kept up, and mustard plasters applied over the heart and around the ankles. If these produce no effect, recourse must be had to blood-letting from the foot or jugular vein, and, as a last resort, an opening into the trachea, through which pure air is forced.

Those overcome by the inhalation of carbonic oxide can be resuscitated only by prompt action and a copious supply of pure oxygen to the lungs.

The Force of the Explosion.—Two volumes of marsh-gas (CH_4) combine with 19 vols. of air and develop 23,550 heat-units, giving a temperature of 6064° F. (6525° absolute) and a pressure of 185.3 lbs. per square inch. If m be the weight and c the specific heat of a gas, the heat required to raise it $t°$ is expressed by mtc. To raise the 2.75 lbs. of CO_2, 14 lbs. of N and 2.25 lbs. of water from 52° F. to $t°$ F. require

$$2.75 \times 0.1711t = 0.470t;$$
$$14 \quad \times 0.1740t = 2.418t;$$

and

$$2.25(160+990)+2.25 \times 0.2675t = 0.602t+2587;$$

whence

$$0.470t+2.418t+0.602t+2587 = 23{,}550.$$

The force of the explosion of two volumes of marsh-gas developing 23,550 heat-units may be ascertained to be nearly 30,000 lbs. per square foot by the following analysis:

Assuming the initial temperature of the marsh-gas and the mine air to be 62° F. or 523° absolute, because it requires 3.886 heat-units to raise the aggregate products of combustion one degree, the degrees to which the final gaseous products of combustion will be raised are

$$23{,}550 \div 3.886 = 6064° \text{ F.}$$

The volume which these products seek to occupy is

$$(523+6064°) \div 523 = 12.6 \text{ atmospheres.}$$

12.6×14.7 per square inch equals 26,671 lbs. per square foot.

The Barometric Relation of Explosions.—An attempt has been made to hypothecate a relation between the periods of gas outbursts and the movements or seasons of low barometer, has failed to show any connection. A falling barometer has not invariably been followed by a heavy discharge of gas, nor does a study of the tables show its unfailing precedence to the evolution. While laying stress on the acknowledged fact that December is the worst month, there appears to be no "off day" for explosions, which are equally abundant on any day of the week. An excessively low barometer at the sea-level is 28.3 inches—a fall of only 6 per cent of the total pressure, and of but 1 per cent, or less, of the pressure of the magazine gas. Upon the emissions from the pores of the coal and from goaves the effect of a barometric depression is noticeable. But even here

an acre of ground of standard thickness will, with a barometric fall of 0.1 inch, exude only 18 cubic feet of mixture for every 25 yards of length of face exposed.

A tabulation of the barometric variations with reference to mine explosions was made in Westphalia, during 1896, with the result that of the 42 explosions recorded 41 are attributed to fire-damp alone, while in one of them coal-dust participated; out of the total number, 27 happened when the air pressure showed a tendency to fall suddenly or was at its minimum, while in the 15 others the air pressure was at maximum, or showed a tendency to rise. As regards the places where they occurred, the explosions are divided into eight in exploring or preparatory workings in rock, 26 in preparatory workings in coal, and eight in the actual getting of coal, while the gas issued slowly in 28 and suddenly in 14 cases.

The Diffusion of Gases.—It is fortunate that the gases evolved from coal or produced by the various processes of decomposition, combustion, and exhalation do not accumulate in separate layers in the workings, with the heavier gas at the bottom and the lighter one near the top, except in abandoned places where the air is allowed to stagnate; but instead of this even a little circulation will set up an individual motion of the separate particles of the gases, by which they become gradually diffused throughout the mass until, after sufficient time has elapsed for the purpose, they are found intimately blended, whatever may be their relative densities. This is not a chemical mixture, but a purely mechanical blending, depending upon the relative tensions of the gases. The rapidity of this diffusion into atmospheric air is inversely proportional to the square root of the density of the penetrating gas. Marsh-gas, therefore, mixes most readily with the air, carbonic oxide not quite so readily; sulphuretted hydrogen less so, and carbonic acid making with difficulty an intimate mixture with air. This principle of diffusion is an exceedingly valuable one to the safety of the mine and the purity of its air, since the more ready the diffusion of the gas the more easily will the gas be cleared away. Thus, by the creation of an air-current

throughout the workings, the gases are mixed with the circulating pure air, are diluted, and swept out of the mine.

Testing Mine Air for Gas.—The condition of the coal workings is usually examined and tested daily by the fire-boss, one of whose duties consists in ascertaining the degree of saturation of the mine air at every place of work before the men are permitted to enter. The test is made by a candle or safety-lamp, the flame of which gives evidence of the presence of an accumulation of combustible gas. This method requires a skilful, steady hand and considerable nerve. Shading the flame of a candle or lamp with one hand, and raising it upward, the fire-boss watches the behavior of the light. If any inflammable gas is present, the flame elongates and becomes smoky. In this event the test ceases, the flame is lowered, and the fire-boss withdraws. The face of the coal or the room showing these symptoms of danger is then supplied with more air, the employees being meanwhile barred from entry to the place.

The Height of the Flame in Gas.—A gas gives evidence of its presence upon a flame immersed in it by the elongation of the flame, surrounded by a blue nimbus or aureole. The more volatile the illuminating-oil used in the lamp, the more sensitive is the flame to the presence of these gases. Thus naphtha, benzine, and kerosene, in the order named, are far more sensitive indicators of gas than is the heavy lard-oil.

The Height of the Cap on the light of an ordinary safety-lamp depends upon the percentage of gas to that of air in the mixture. By observing the height one may determine the approximate percentage as follows: Divide the constant 94,000 by the height of the cap in eighths of an inch and take the cube root of the quotient. Thus, when a blue cap is found in a mixture under test to be 2 inches in height, then the cube root of 94,000 divided by 16 equals 18. In other words, there are 18 parts of air to 1 of gas. If, in another test, the length of the blue cap be only $\frac{1}{8}$ of an inch, then there will be 46 parts of air to 1 of gas.

Testing Lamps.—Ordinary safety-lamps do not reveal the

presence of a quantity of gas less than 2 per cent. The Hepplewite-Gray is more sensitive than the unbonneted Davy or Clanny. It burns benzoline, and shows a cap ½ inch high in the presence of 1 per cent of CH_4. The Pieler spirit-lamp is always a good gas-tester, which in air containing ½ per cent of combustible gas will give evidence of it in a cap 1 inch high. The Wolf safety-lamp, burning naphtha, shows a very conspicuous halo when placed in a mixture of fire-damp and air; in a mixture of ½ per cent of gas the flame is 1½ inches in height, and in 2½ per cent of gas the flame is broader, and may even be extinguished. The Beard-Mackie lamp carries a graduated scale making the height of the cap visible. A bent inverted U rod has platinum wires stretched across the arms, at suitable intervals, which become luminous by contact with the flame. The highest wire furnishes the guide to the test.

The Shaw Gas-testing Apparatus recognizes the presence of explosive gas and is used for an approximate test of mine air. It meets with favor where the exact analysis of the mine atmosphere is not required. While the apparatus is capable of demonstrating the quantity of explosive gas in the atmosphere, it is incapable of distinguishing between them, and thus fails to furnish any clew as to the variety of gas therein contained. Of other detectors, those depending upon the difference in density of the gases are unreliable, because changes of temperature will produce similar results. Aitken's indicator is ingenious, but not much more reliable. Its thermometer is coated with platinum-black and plaster of Paris, and when exposed to fire-damp becomes heated. If the difference of temperature between it and the normal air always bore a comparable ratio to the percentage of fire-damp contained, it would work well. The special forms of gas-detectors do not serve for illumination.

The Amount of Air Required for Combustion.—In attempting to specify the amount of air required for proper ventilation of a mine, we are treading upon uncertain ground. Within close limits we may ascertain the amount required for the vital chemical purposes of horse, light, and man. A pound of carbon

requires for complete combustion $2\frac{2}{3}$ lbs. of oxygen, and produces $3\frac{2}{3}$ lbs. of CO_2. Hence the ordinary-sized mining-candle burns up 11.8 cu. ft. of air, and discharges 3 cu. ft. of CO_2. Eminent medical authorities state that a man consumes about 1 cu. ft. of air per minute, converting the life-giving principle into 2.1 cu. ft. of CO_2 per hour. The respiration of a horse is about 13 cu. ft. CO_2 per hour. The deflagration of a pound of explosive produces about 2.6 cu. ft. According to Angus Smith, two hewers using a $\frac{1}{2}$-lb. candle and 12 ounces of powder produce $25\frac{1}{2}$ cu. ft. CO_2 in a shift.

The amounts of air sufficient to satisfy the conditions of combustion during the generation of the respective amounts of CO_2 are small, and if the exhalations were instantly removed, the theoretical chemical supply would suffice. But the air in the confined spaces of mine-workings is somewhat stagnant, and the atmosphere is further deteriorated by the exhalations from man and beast. Some of these are not easily detected chemically, but are more deleterious than CO_2, which is not the sole test of vitiation.

The Amount of Air Required for Ventilation.—The hot, noisome emanations, the poisonous exhalations, the unconsumed azotic gases, and the exuding pent-up gases from the coal must be rendered comparatively harmless. This requires a large volume of air for their dilution and renewal. Pure dry air contains, by volume, 21 per cent of oxygen, O, and 79 per cent of nitrogen, N; and every 1000 cu. ft of it, weighing nearly 81 lbs., contains only about 18.7 lbs. of the life-supporting constituent, the remainder being matter inert in its physiological effects.

To furnish $2\frac{2}{3}$ lbs. of oxygen for a pound of carbon would require 142.2 cu. ft. of air for combustion alone. To dilute the carbonic acid produced to a wholesome degree, it will require 2260 cu. ft. for each pound of carbon.

Judging by the rough test afforded by the sense of smell, the air of a room ceases to be wholesome when it contains more than 6 parts of CO_2 in 10,000. And to preserve the lowest standard

tolerated by sanitarians, 1 in 10,000, the supply will be proportioned as follows: 59 cu. ft. per hour per light; 4585 per horse; 9192 per pound of powder; and 1500 per man employed. Competent writers vary in this matter, and the statutes of the various States differ in their requirements (55 to 300 cu. ft. per man per minute). But the allowance for a mine cannot be based on the single per-capita element, for it will be seen later that the friction or "drag" of air, in moving through headings and along faces which increase with the developments, diminishes the volume of air actually allowed to move. Moreover, the emission of gas from the strata, proportional to the area exposed and the character of the coal, constitutes another and constant source of pollution. In all preparatory and prospecting work in virgin ground an extra allowance of fresh air is necessary. Cognizance must be taken of this unfailing source to the extent of an hourly allowance of 0.3 cu. ft. of air per square foot of working face, in a dry, dusty, fiery mine. For a non-gaseous seam 0.1 cu. ft. will suffice. Some property-owners allow also 200 cu. ft. of air per hour for every acre of goaf. For the eruptions from the magazines no provision can be made except vigilance and discipline.

The Water-gauge (Fig. 154) consists of a U tube whose arms contain water and are provided with measured scales graduated to inches above and below the normal of the water in the columns. If a gauge be inserted through the stopping (Fig. 155) separating the bottoms of two ventilating-shafts of the mine, the water will remain at a normal level, if the temperatures and tensions of the gas and air in the two shafts be equal; but if by any cause the tension or temperature be changed in either shaft, the ensuing difference in pressures will be communicated to the connecting-arms of the water-gauge in such manner that the cool or denser air will force down the water column in the arm on its side of the stopping and elevate the water column on the opposite side. This difference in level, m, is read on the attached scale and represents the motive column, M, which is capable of producing motion. If, now, this excessive pressure may be allowed to expend itself in producing motion through the

workings of the mine, in circulating the air which ultimately is discharged through the lighter column of upcast, the level of the water in the gauge will fall slightly until equilibrium will be established, when its difference in level will represent the difference in pressure, p, at the bottom of the two shafts on either side of the stopping. The excessive pressure, P, is expended in doing work of two kinds: (1) in overcoming

FIG. 154.—Water-gauge. FIG. 155.—The Position of the Water-gauge on the Door.

the friction to the passage of the current of air through the workings, from one shaft to the other, and (2) in creating motion. The latter work is measured by the velocity of the outgoing current, the former is measured by the height of the water in the gauge, Fig. 154, or the manometric depression, m. Its difference in level constituting the water-gauge reading, measures the force which is required to drive the air through the mine. It measures the loss due to friction or the "drag" of the mine. Be the quantity of air large or small, it gives no measure of that volume, but, paradoxical though it may seem, only of the power of the ventilator.

The Mine Resistance.—The resistance of the mine is a definite quantity, and bears no relation to the capacity or qualities of the ventilating appliances or methods. The water-gauge, therefore, which measures this resistance is a "function of the mine," and by it may be determined the relative efficiency of the mine to pass air through its ways. The water-gauge reading in the

majority of mines varies between 1 inch and 3 inches. Few have a larger resistance. The mine with airways of large cross-sectional area and with a well-distributed current should have a water-gauge reading, or resistance, not exceeding 1 inch for each hundred thousand cubic feet circulating through it. The mine having a larger ratio of water-gauge reading than this either has airways of insufficient size or is not receiving the current properly regulated.

The Equivalent Orifice of the Mine.—The resistance of the mine to the passage of an air-current is often expressed in the term, the equivalent orifice of the mine. By this is understood the area of a thin orifice which offers a resistance to the passage of a current of the same volume, Q, equal to that which is circulating through the mine. The equivalent orifice of the mine, A, bears a certain ratio to the quantity, Q, and to the water-gauge reading, m, which is variously expressed by different authors, the general formula being

$$a = CQ \div \sqrt{m},$$

the value for C varying between 0.00037 and 0.00066. That usually taken in calculations is 0.0004.

The equivalent orifice of most mines varies between 10 and 100 sq. ft. Inasmuch as water is about 833 times as dense as an equal volume of air, the column depressed in the water-gauge corresponds to a height of $833m$ inches of air at 62° F. and 30 inches barometer; the head, M, of air measured in feet, to which the manometric depression is due, is

$$M = 69.4m.$$

The value for the corresponding differential pressure, P, in pounds per square foot, is

$$P = 5.184m.$$

EXAMPLE.—How does the efficiency of a mine carrying 178,000 cubic feet of air compare with another having 284,800 cubic feet per minute? The water-gauge readings are 2.7 inches and 3.4 inches, respectively.

$$a = \frac{0.0004 \times 17,8000}{\sqrt{2.7}} = 43.36 \text{ square feet;}$$

$$a = \frac{0.0004 \times 284{,}800}{\sqrt{3.4}} = 61.78 \text{ square feet.}$$

Their relative efficiencies are 0.70184:1, respectively.

The following table illustrates the rate of decrease of water-gauge in two mines, respectively, carrying 30,000 and 100,000 cu. ft. of air per minute for various equivalent mine orifices.

Equivalent Orifice, Square Feet.	Water Gauge, Inches.		Equivalent Orifice, Square Feet.	Water Gauge, Inches.	
	30,000 Cubic Feet Air.	100,000 Cubic Feet Air.		30,000 Cubic Feet Air.	100,000 Cubic Feet Air.
5	2.56	....	25	0.23	2.56
10	1.44	16.00	30	0.16	1.78
15	0.64	7.11	40	....	1.00
18	0.45	5.00	50	....	0.64
20	0.36	4.00	100	....	0.16

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CHAPTER XIII.

METHODS OF VENTILATION.

The Ventilation System.—To obtain circulation through the mine, a conduit must be furnished by which the warmer and lighter air may ascend to be supplanted by cold or compressed air entering by a different compartment; and to maintain a constant air-current throughout the workings, both inlet and outlet must be afforded for the air by means of two separate entries or by partitions in the one shaft.

The Ventilation of Single Entries.—Shafts, in process of sinking, or a mine having but a single entry, may discharge their vitiated air through the wooden air-tight box-pipe provided for the purpose, or, if there is small liability of corrosion, through a galvanized-iron pipe, the remainder of the entry furnishing the inlet. Because of the wide difference in the areas of the two airways so provided, the ventilation is not likely to be good, and it is far better to divide the main tunnel or shaft or mine-working into two compartments of nearly equal area, one of which will serve as an outgoing conduit.

From the fact that the current in a single-entry mine is continually interrupted by the other uses to which the compartment is put, and that there is a liability to injury of the partition, box, or pipe, this plan is objectionable when a large volume of air is required, because the safety of a great number of men is dependent upon this airway for escape. The wind, moreover, disturbs the ventilating current; the movement of cars, cages, and rock or coal in chutes is also irregular in its influence upon it; and the unusual heat from underground steam-pipes, engines, etc.,

sets up counter-currents, though any of the causes mentioned may occasionally have a beneficial effect. Thus a double entry to the mine becomes not only precautionary, but also imperative; and as the depth and extent of workings increase, the insufficiency of a single entry becomes more and more manifest. Even metalliferous mines should be provided with a double entry, for the numerous caves that have occurred, penning in dozens of men without chance of escape unless the rescuers can reach them before suffocation ensues, and the fires that frequently cut off the employees from the outlet and suffocate them before extinguishment is effected, are sufficient arguments in favor of double entry, even if the necessities for better air do not appeal to the operators.

Ventilation by Double Entries.—All collieries have two outlets, separated by a safe distance of unbroken rock. The upcast, advisably, should terminate in a large chimney, high enough that its draught be not influenced by changes of wind or the surrounding buildings. The location of the two entries, in reference to each other, varies within wide limits. One plan consists in having them near together, thus concentrating the plant. Both airways being carried with the development, the current passes through to the extreme end of one and returns by the other. Then as the work deepens, each lower lift is connected with the airways of the upper lift, and receives ventilation with its advance. The other plan is the "diagonal system," the shafts being at the extremities of the workings. While this is well enough for the long-wall method, the ventilation must meanwhile suffer until the connection has been made.

Two compartments in a single entry may be easily obtained in coal furnishing sufficient rock from the roof or from partings by driving a wide gallery and walling it up centrally with the waste; but if there is not rock enough for this, two entries are carried, with the usual pillar between them, having connecting "throughs" at intervals of less than 100 feet, each being closed as fast as the next one is completed. To ventilate that part of each entry between the last connection of the entries and its

face, it is subdivided by a canvas brattice fastened at the near side of the "through" and leading up to the work. On either side of this the current flows. The faces may be connected by pipes through the door closing the intake entry without interfering with the haulage. The practice of relying upon diffusion to do the work of ventilation is pernicious. These remarks also hold true regarding the "throughs" connecting the rooms in pillar and stall workings, where diffusion is usually relied upon for the needful amount of oxygen.

Planning Airways.—A large number of the coal-mines depend for their ventilation solely upon natural means, and this may suffice in small mines. But as the workings are extended, the numerous connections which are necessary for development or convenience of handling the materials may be planned to serve also for ventilating ways without additional cost.

In planning the direction of gangways and of rooms in coal-mines, usually the question of haulage is of the first consideration, unless it be that the "cleats" are so pronounced as to determine the direction of work. At the same time due attention must be given to the matter of ventilation, that the requisite amount of air be given each working-room, and that too many men be not dependent upon the same air-current circulating through the mine; whenever the mining conditions require a subdivision of the incoming air-current into small currents, each being distributed to its own district and group of men and each separately discharged, it becomes evident that the ventilation of such gaseous mines must receive special attention, not only as to the direction in which the airways are driven and their cross-sectional dimensions, but also as to the means of producing the supply of air. In such cases the fresh air should be carried, if possible, to the deepest point in the mine, whence an ascending current may be conveyed through the workings until it is returned to the surface. Especially is this advisable in steep coal-seams carrying fire-damp.

The ventilation must be so arranged that as many independent ventilation districts as possible be provided with sepa-

rate air-currents; and especially must each lift of workings be supplied by the shortest way with the necessary quantity of fresh air, while within the separate lifts of workings the air-current must always be ascending—except in cases in which the descending air-currents are not used for any further ventilation purpose, or when, in certain well-ventilated working places, excessive thrust of the measures renders very difficult the keeping up of special return airways.

In metal-mines, where the development is of slower growth, the rock hard, and a comparatively few men are at work, the amount of air required is small, either for inhalation or for the dilution of the gases developed therein; hence a single shaft with two compartments may suffice, the circulation being left to natural sources. This, however, will be inadequate when the shafts and workings reach a depth of several hundred feet, in which case other means must be employed. The use of compressed air for drills, pumps, etc., may supply the deficiency of pure air which natural ventilation may fail to furnish, yet a fan, exhausting the air from one outlet or forcing the air into the other, seems imperative with extensive workings.

The Underground Temperature.—Below the depth where atmospheric changes have no influence, the temperature of the undisturbed rock increases with every increase in depth. The depth at which the temperature of the ground will be found to be invariable and equal to the natural temperature of the locality is about 50 feet below the surface. Beyond this it is an observed fact that in all artificial openings the temperature of the rocks increases for at least a moderate depth, within which the mine operator is concerned, at the rate of about 1° F. for every 68 feet of depth. This increment is not constant for all localities, nor indeed for the same mine, but generally it may be said that as we go down the temperature of the mine increases more or less uniformly. This increased heat is often a great drawback to mining, and will ultimately limit it apart from the lesser mechanical difficulties. As to what would constitute the limiting depth to which mining may be prosecuted, it can but be said that at

present several mines, with the exception of the Comstock and those which are in ore-bearing districts feeling the effects of solfataric action, are working at over 4000 feet. Regarding the exceptions stated, it is certain that unless some means be discovered for rendering their lower levels habitable, the limit of mining depth is soon reached. It is stated that a 2800-foot level of the Yellow Jacket Mines has been abandoned because of the excessive temperature, in many rooms of which the miner is compelled to return to a cooling station after laboring only twenty minutes.

An interesting report bearing upon this question of the rate of increase of temperature with the depth of subterranean explorations, made by a sub-committee of the Royal Commission on Coal, reaches the following conclusions: That the limit of depth to which mining is possible depends upon human endurance of high temperature, and upon the extent to which it would be possible to reduce the temperature of the air which comes in contact with the heated rocks; that there is no limit caused by considerations of a mechanical nature as to the size of rope for hoisting-engines, nor by any consideration of the enhanced expenditure for shaft sinking, for haulage, or for pumping. Regarding the latter, the experts testifying before them demonstrated that water is seldom, if ever, met with in large quantities at great depths in mines. It therefore appeared that this increase in temperature is the only element needing consideration regarding the limits of prospective sinkings or workings.

A summary of the results of temperature observations made under the direction of the British Commission Committee shows the mean increase of temperature per foot to be 0.01563, or 1° F. in 64 feet, the extremes being 0.0077 in the Bootle water-works bore-holes, and 0.025 in the Carrickfergus shaft. At the Adelbert shaft, Prussia, observations five times a month, in different levels, for a year, could deduce no regular law of increase; at the 30th level, 3200 feet, the temperature was 98° F.

Natural Ventilation.—The temperature of the air inside the mine differs from that outside. The mass which is the warmer

will rise, enabling the colder mass to fall. Circulation is established so long as this difference exists. So that, if two openings be made and connected below, a current will be established down the lower and shorter opening in winter, and up the same during the summer, as the arrows (Fig. 156) marked S indicate. In winter the direction of the current follows that of the arrows, W. The amount of air thus set into circulation by the changes of the exterior temperature will depend upon the relative difference of temperature between the mine and surface, and also upon the depths of the shafts. When these differences are slight it is not easy to predict the direction which the current will take. As, for example, in the fall and spring it will fluctuate from one to the other. When, however, these differences are great, a current will be set up which tends to continue in the same direction so long as these differences remain. Thus in summer the current will follow (Fig. 156) the arrows, S; in the fall little or no current will be set up; in the winter the current will reverse and follow the arrows, W; in the spring the conditions are again nearly balanced, and little current will flow. When the shaft attains a depth of 800 feet, the subterranean air is always hotter and lighter than the surface air at any season; and unless the two outlets have a great difference in elevation, an uninterrupted current will continue, without fear of reversal, down the lower and shorter opening.

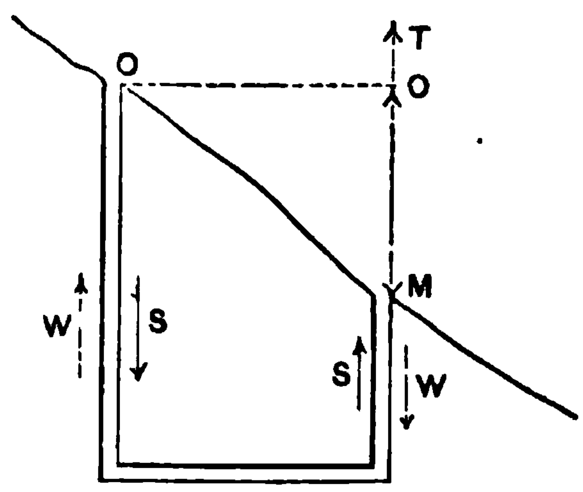


FIG. 156.—The Circulation of the Currents in Summer and Winter.

While this method may be universally practised under favorable conditions in metal-mines, it is evident that in collieries one danger arises from the reversal of current, for at one time the current, following the arrows marked S, carries the air through the gangways, whence it is distributed among the work-rooms, to be returned to the surface by way of the longer and deeper shaft; but during the other season the air may follow the arrows marked W, thus entering the working places first, and

departing thence through the gangways, making its exit by the lower or shorter shaft. If, now, there be a number of abandoned rooms or goaves connected with the working-rooms, it is manifest that in the latter season the air must pass through them first before reaching the men at work, and thus carry noxious gases with the current to spread calamity by explosion or fire. Again, the fact that no air will circulate during the vernal seasons would render the provision for supplementary means of ventilation imperative. Air-currents which have served for ventilating preparatory workings or prospecting drifts in the virgin seam never should pass over stalls or working places where men are engaged, on its way to the air-level.

The Tension of the Atmosphere.—The air possesses, in common with all other gases, in consequence of the repulsion between its molecules, a tendency to expand into a greater space. This indefinite expansion, by reason of which every gaseous fluid, not restricted by an extraneous force, continues to expand to the tenuity of interstellar space, results in the creation of an air-current whenever by an increase of temperature or a diminution of pressure the given mass of air expands in opposition to the attraction of the earth and rises into the upper strata. This upward flow will continue so long as the gas expands until the resistance encountered by it is equal to, or greater than, the repulsion among its molecules. It is this readiness with which gases tend to adjust themselves to the varying conditions of temperature and pressure that plays so important a part in mine ventilation. The tension of a gas increases with its compression, and the density of a given mass of air is proportional to its tension.

The Weight of Air.—This may be calculated for different temperatures and barometric pressures by the following formulæ:

The volumes, u, assumed by a given weight of a gas are inversely as the corresponding pressures per unit of surface, provided the temperature remains constant:

$$u : u' :: p' : p.$$

If the temperatures change while the pressures are constant,

the volumes, reduced to absolute zero ($-461°$ F.), will be found to vary proportionally.

$$u : u' :: 461 + t : 461 + T;$$

t and T being, respectively, for up and $u'\,p'$, Fahrenheit readings.

The weight of a cubic foot of air at a temperature t and a barometric pressure B, in inches of mercury, is obtained by the following formula, and at a temperature T, is W, expressed as follows:

$$w = \frac{1.3253B}{461+t}, \quad W = \frac{1.3253B}{461+T}.$$

A table of weights of air for various temperatures and a constant pressure, B, will be found in Chapter IX.

The Production of Draught.—The energy in a mass of air compressed to a certain degree may be measured by the work restored by it in expanding, and this energy may be converted into motion producing a current, or it may result in a pressure when that tendency to motion is resisted, or when the motion is suddenly arrested.

That portion of the energy stored up in the air which is expended during its expansion in dynamic effect causes a "wind" or "draught," the velocity of which depends upon the difference in tension.

Whether the difference in tensions is produced by change of temperature, or of pressure, the velocity acquired by any gas is

$$v = \sqrt{2gH},$$

in which H is the head due to the difference between the tensions, or densities, of the initial state of the cool or compressed gas and the final state of the hot or expanded gas. Atmospheric air, at a barometric pressure of 29.92 inches, at a temperature of 32° F., when flowing into a vacuum, attains a velocity, in feet per second, which is equal to

$$v=\sqrt{2gH}=8.02\sqrt{\frac{P}{W}}$$

$$=8.02\sqrt{\frac{14.7\times144}{0.00118}}=10758.9 \text{ feet.}$$

The total difference of pressures per square foot is represented by P in pounds, and the weight of a cubic foot of the warmer or attenuated gas by W.

So, too, the velocity with which compressed air or steam escaping freely from a pipe or other reservoir of the same may be ascertained, the value to be supplied for H, the head to which the velocity will be due being equal to the pressure in pounds per square inch under which the gas exists, multiplied by 144 and divided by the weight in pounds per cubic foot of the exhausted fluid.

The Motive Column.—This is the head producing motion. When, however, two masses of air of equal height but of different tensions, p and p', are exerting a pressure upon one another through a connecting conduit, the resulting difference in pressure per unit of area of base measures the motive force, in which case P is the total difference of aerostatic pressure in pounds per horizontal square foot of sectional area of base, and W the weight per cubic foot of the rising column of air.

If, then, a column of air at t° F., D feet high, with a base of one square foot, be heated to T° F., its new height would be greater by some quantity, which we may call M. If two such columns be connected, being of the same depth but of different temperatures, t and T respectively, the latter column would be lighter than that at t° by a quantity $D(w-W)$; and so long as this difference in temperature is maintained, this difference of pressure, which we may represent by P, ensues, by reason of which the hot column of air would be driven upward, producing a draught with a velocity, V, due to the aerostatic head, M. To hold this force, P, in equilibrium would require a resistance $D(w-W)$ pounds per square foot; or the pressure of an addi-

tional column of warm air weighing W pounds per cubic foot, of a height of M feet,

$$M=\frac{D(w-W)}{W}=\frac{P}{W}=D\frac{T-t}{461+t}.$$

This quantity M is known as the motive column to which is due the velocity of the flow of air, and if no resistance is offered to it, motion will take place. It may be represented by OT (Fig. 156), which equalizes the pressure of the unequally heated columns of air below the level of the line OO.

When two such shafts are of unequal depth, as at O and M, Fig. 156, and have equal exterior and interior temperatures, a rarefaction of the air in either one of them not affecting the other would result in a diminished pressure upon the bottom, just as is obtained by a difference in temperatures; a rising current is established therein, with a velocity dependent upon the ratio $\frac{P}{W}$, in which P is the difference in the weights of the two shaft columns of a base one horizontal square foot in area and a height D; and W is the weight of a cubic foot of the rarefied air.

For the purpose of mine ventilation there will be required a motive column much larger than that here obtained, because of the enormous friction of the air in rubbing along the rough surface of the workings, turning sharp corners, and squeezing through small openings. The resistance due to this cause amounts often to as much as 90 per cent of the power. In other words only one tenth of the theoretical motive column becomes effective in producing a current, and the actual velocity of the air-current, v, does not exceed one third of the theoretical velocity, V, due to the head, M.

The principle upon which chimney draughts for boiler or other heating apparatus depends is also similar to that here described, excluding, of course, frictional allowance. In chimneys for boiler-furnace draughts the fire burns best when W is

$0.5w$, and the height of the manometric column in the chimney is about one half an inch of water.

It is evident, therefore, that the height of a motive column depends upon the difference in temperatures or a difference in tensions, or both, of the gaseous mixture contained in the two shafts or entries to the mine. A measure for this motive column may be had in feet of head of pressure per square foot of area of the base, or in the number of inches of a water column in the manometer which corresponds to this weight. Insomuch as a column of water 1 inch high and a square foot in area of base weighs 5.184 lbs., the height, m, of a water-gauge column which will balance the pressure P is equal to $P \div 5.184$.

Let M be the head corresponding to the motive column, v the velocity of flow of the upcast air per second; then is $\frac{v^2}{2g}$ the effective velocity head of the issuing air; and if W is the weight of a cubic foot of the warm or attenuated rising air, the theoretical energy of the moving air per second is WM, and the effective or actual energy is $W\frac{v^2}{2g}$, or $0.01553Wv^2$. That portion of the energy which is consumed in overcoming the friction of the mine is therefore $W(M - 0.01553v^2)$. It is this lost energy which is measured by the water-gauge. As the mine resistances are reduced, so the water-gauge reading is reduced, and the efficiency of the mine increases, permitting a greater actual return from the expenditure of the same potential force.

The height, m, of the water column being measured in inches, the number of horse-powers, H, necessary to produce a ventilating current of Q cubic feet per minute is known by the following formula:

$$H = 0.0001571Qm.$$

The indication, therefore, which the water-gauge reading gives of the ventilating force is evident in the above formula—that for a given quantity of air, Q, in circulation, the horse-power necessary to produce ventilation increases with the resistance of the mine.

The Systems of Producing a Ventilating Current.—The several methods of accelerating the natural ventilation and distributing air properly throughout the mine contemplate some system either of decreasing the tension of the mine air to enable the return current to ascend to the surface, or of increasing its tension by the use of a compressor to force atmospheric air into the mine. The several means by which these results are attained may be, first, a furnace built at the bottom of the outlet shaft, or a fan; or, second, a blowing, propelling, or air-compressing fan at the mouth of the inlet shaft. By either of these methods a different state of tension is produced in the two shafts connected below, and, in the effort to establish equilibrium, the air is set in motion, a draught is created, and a current is established that flows through the air-courses at a velocity dependent upon the head due to the difference in pressures, as has been seen on the preceding page.

Furnaces are employed for increasing the temperature and are constructed in such manner as to be remote from direct contact with the coal, yet in close proximity to the shaft which constitutes the outlet for the mine air, and in a gallery through which circulates air from the workings. The pit selected for the outlet should be that one which would naturally carry the flow in winter. The furnace is simply a fireplace, walled and roofed by a fire-brick or common-brick arch (Figs. 157 and 158). When special care is taken, a second wall is built outside and over, with an air-space between, to isolate it from the coal and prevent fire. If the roof is wet, a double arch must surmount the furnace, as otherwise the steam generated will burst the arch. If the mine is fiery, or considerable dust is floating, care must be taken that the gases are well diffused, or else the current must not be brought into close proximity with the fire. In such cases the current is split, a small portion being heated over the fire, the remainder passing through a "dumb-channel," entering the upcast 50 feet or so above. A still safer plan passes all of the fiery current through the channel, and feeds the furnace by a split current of fresh air direct from the intake.

The size of the grate depends upon the work to be done. Its bars are 3 feet from the floor, slanting upward toward the

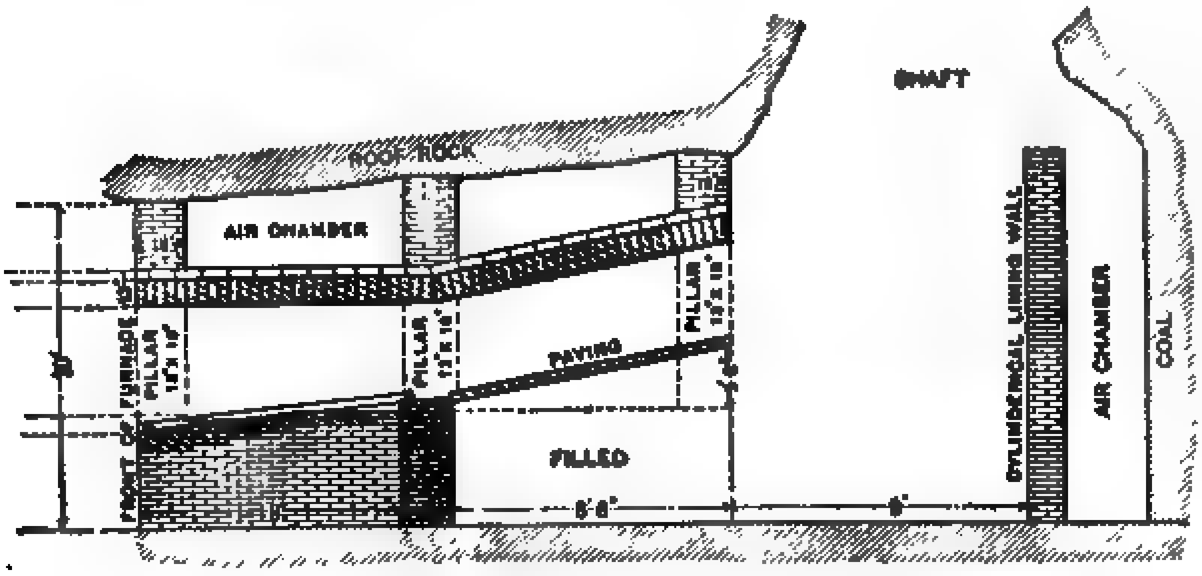


FIG. 157.—Longitudinal Section of a Furnace.

shaft 1 to 6, distance to the roof 4 or 5 feet. The width, wall to wall, is 6 feet and its length from 4 to 12 feet, according to the volume of air to be moved, which is about 1500 cu. ft. per square foot of fire surface on a properly constructed furnace.

FIG. 158.—Cross-section of a Furnace.

An ordinary furnace of 34 sq. ft. heating-surface, costing $130, will heat a column of air such as will furnish 29,000 cu. ft. per minute. A large number of furnaces 10×12 furnish 200,000 cu. ft. The cross-sectional area must be 50 per cent greater than the upcast airway, and the shape capable of regulation by

double-sliding iron doors, to produce varying degrees of contraction and of combustion. The fire is spread over its entire width, and over only as much of its length as is necessary to furnish an adequate motive column, at a temperature of 140° to 160° F. Emergencies, as low barometer and high thermometer, and the cleaning of the grates, require other and more heating-surface. The coal consumed is 2 to 5 tons per day, spread thin and evenly over the bars, and fed from both ends, on a long furnace. This rate is 40 to 70 lbs. per hourly H.P. of work done on the air. Attendance, etc., is $5 per day.

Q being the quantity of air in cubic feet per minute, W the weight of a cubic foot of return air, T being the temperature, F.°, of the up-cast air, and t that of the air in the return airway, the number of pounds of coal consumed by the furnace per hour is

$$x = 0.00376WQ(T-t).$$

The area, F, of the grate-surface in square feet is about one tenth of the hourly coal consumption, in pounds, and its relation to the depth, D, of the furnace below the surface is known by the expression

$$FQP = 1,716,000\sqrt{D},$$

P being the manometric depression in pounds per square foot, and Q the volume of air per minute.

The Limits of Furnace Ventilation.—A method so simple and cheap in construction and easy of management presents advantages which have long commended it to mine operators; nevertheless the difficulties with its use, the dangers which attend the exposing of an open fire in gaseous districts without the possibility of introducing a safeguard, the numerous calamities traced to the furnace which has fired either the solid coal surrounding it, the gases in the return air, the timbers of the shaft, or even the surface plant, and its lack of economy in shallow pits, were soon made manifest. The atmospheric changes of the seasons reduce its efficiency, a decrease in the barometric

pressure and an increase in the surface atmosphere reduced the action of the furnace, and notwithstanding its great superiority over many other mechanical appliances, it has gradually been supplanted by fans. The power of the furnace increases arithmetically with the temperature, and that with the amount of fuel burned. The quantity of coal that can be consumed upon a given area is limited. The resistance of the mine increases on the other hand geometrically with the square of the velocity of the current, and it is therefore manifest that between the several conditions the furnace limit is soon reached. Many furnaces may be cited supplying to the mine over 200,000 cu. ft. of air per minute; and enormous as they are, their cost is very little less than that of a modern fan of large size; but when we contemplate the huge pile of coal thus consumed for the production of the current, we are forced to the conclusion that efficient furnace ventilation is a luxury which the coal trade cannot long endure. Perhaps as the depth of the collieries increases to about 2000 feet the furnace may be reinstated.

With the atmospheric air at 62° F., and the furnace-heated air at 132° F., the water-gauge depressions, *m*, produced at various depths of furnace are as follows:

D.	*m.*	*D.*	*m.*
50 feet,	0.086 inch.	1000 feet,	1.735 inch.
400 "	0.694 "	2000 "	3.471 "
700 "	1.215 "	4000 "	6.943 "

These are in accordance with the formula for estimating the manometric depression:

$$m = D \frac{W}{5.184}\left(\frac{T-t}{461+T}\right).$$

EXAMPLE.—A colliery has two shafts 1000 feet deep, 12 feet in diameter; temperature in the downcast is 60° F.; barometric pressure is 30 inches. 150,000 cubic feet of air are supplied per minute by a furnace. Required the temperature of the upcast and the horse-power necessary to produce the ventilation, the mine being supposed to show a water-gauge resistance of 2 inches of water. 520° F. and 163.5 H.P.

Assume the coefficient of friction for the smooth shafts to be as great as that of the rough mine galleries; then

$$p=\frac{0.0000000217\times1000\times37.7\times(150{,}000)^2}{(113)^3}=12.8.$$

Each shaft therefore offers a resistance equal to 12.8 lbs. per square foot. The total resistance then is $25.6+10.368=35.968$. The work done in ventilating is $150{,}000\times35{,}968=5{,}395{,}500$ ft.-lbs., or 163.5 horse-power.

The temperature of the upcast shaft is

$$\frac{P}{W}=\frac{(461+T)35.968}{39.759}=1000\frac{T-60}{521}.$$

Or, by another method: A cubic foot of air at 60° F. and 30° barometer weighs 0.0766 lb. 150,000 cubic feet of the circulating air weigh 11,490 lbs. Since the furnace is performing 5,395,500 ft.-lbs. of work upon 11,490 lbs. of air, the height through which it is moved is 470 feet in one minute. Then M is 470 ft., $t=60°$ F., and $D=1000$ ft.

$$T=\frac{M(461+t)}{D-M}+t=520°\text{ F.}$$

From this it is seen that the temperature of the upcast air necessary to force 150,000 cubic feet of air through the mine is dangerously high. The furnace must be replaced by the exhaust-fan, or the frictional resistances must be reduced by enlarging the entry-ways.

What should be the size of the airway shafts in the above case, that the upcast air be not hotter than 190° F.? By substitution above, M is found to be 200 ft.; the work is then $11{,}490\times200=2{,}300{,}000$ ft.-lbs. (70 H.P.). This value requires that p should not exceed 15.3 lbs., which limits the shaft's resistances to 4.932 lbs., or 2.466 lbs. each. In order to obtain so low a friction the areas are enlarged to a radius of 16.66 feet.

$$pa^3=flmq^2,\quad\text{or}\quad p(\pi r^2)^3=fl(2\pi r)q^2.$$

Types of Fans.—Of the various mechanical ventilators tried in mines, fans remain our main reliance at the present time. As the furnace has in the past supplanted various mechanical devices in the form of pumps and trompes, so fans built on various principles have succeeded the furnace and the steam-jet. There are two classes or types of fans: (1) blowers, either rotary or reciprocating, and (2) fans, propeller or centrifugal. Those of one type sweep out a fixed volume of air at each revolution

and are known as the definite-volume exhausters, under which head come the Root, Baker, Lemielle, Cooke, and Fabry. In the other class, acting centrifugally upon the air, we have a simple revolving wheel always working in one direction, producing by its rotation a pressure or a rarefaction the degree of which depends upon its speed. Of these we have the Guibal, Waddle, Walker, and Schiele ventilators.

The Trompe is a simple application of the injector principle,—water falling in the cylinder and carrying with it air, creates a small intake draught. The volume of air, compared with the quantity of water used, is so insignificant that, unless an especially favorable means be provided for carrying off the water, the ventilation is too expensive to be continued except as a temporary expedient.

Pressure Blowers, either rotary or reciprocating in their action, are of general use in America, being represented by the Root, Baker, and Champion on the one hand, and air-compressor and other reciprocators on the other. The blower forces the air through the intake compartment of the mine, which discharges it at the upcast. These blowers or force-fans are much in vogue for small workings and as expedients in furnishing a separate ventilation for stopes and drifts; but few are employed in coal-mines to produce the total ventilation there required. In metal-mines, however, they are largely depended upon, though they supply a pressure higher than that ordinarily required to overcome the resistance of the mine. They produce air by reason of their high speed at a pressure often attaining 10 lbs. per square inch, whereas a mine requires an initial pressure only sufficient to overcome its resistance, which is rarely greater than 10 lbs. per square foot of base. The blower is a small radial wheel revolving freely in a casing and nearly touching its sides. By a central opening on either side the air is admitted to be acted on and set into rotary motion. These blowers may be had in sizes capable of furnishing as much as 16,000 cu. ft. per minute, requiring from 1 to 15 horse-power for their operation. Some of the blowers are capable of a ready alteration from a

blower to an exhaust, or the reverse, which fact recommends them particularly for wide shafts which are liable to freeze during winter. This is particularly advantageous in metal-mines, where it makes very little difference which way the current moves. In collieries, however, as has been seen, this is not feasible.

The Root Blower or force-fan consists of two interlocking impellers revolving side by side in very close connection, without actually touching one another or the enclosing case. They are made of cast iron accurately bored and dressed to a true surface, so that, while practically no air escapes, there is also no internal wear. At each revolution a definite volume of air enters, is enclosed, and discharged either at the top, the bottom, or the side. They are driven by a pair of external gears at a speed ordinarily of from 250 to 500 revolutions per minute. The extremities of the revolving arms of the impeller section are of an acorn shape, or their surfaces are arcs either of true circles or of cycloids.

The Fabry Blower, which resembles the Root, is much used in the north of France and Belgium. Two fans, each having three broad blades arranged radially, are hung in a chamber. They revolve with equal velocities in opposite directions, the blades coming in contact, isolating a quantity of air, and expelling it into the atmosphere. The success of this blower is attributed to the fact that there are no joints in it.

The Baker Rotary Force-fan has inside of its casing three drums, each being an independent casting turning truly and balanced perfectly to insure a steady motion. The upper drum, which receives the power from the engine, does all the work of blowing, while the two lower drums serve as valves to prevent the air from escaping.

Cooke's Fan is a positive machine. An eccentric drum revolves inside of a 12-foot circular case, very close to which is held a swinging shutter that cuts off the entering current from the discharge. The inlet and outlet portion occupies 235° of a revolution. At the Lofthouse iron-mines are seen two of these side by side, the drums being placed opposite each other on the

shaft, so that the revolving mass is balanced, the discharge equalized, and the efficiency raised.

The Lemielle Blower is a species of air-pump, complicated and leaky, producing large volumes under great rarefaction. It consists of a vertical cylinder, within which a second revolves eccentrically; on this latter are two or more vanes, which in one part of the revolution lie close to the shutter, and in another open and expel the air.

The reciprocating blowers have been displaced almost entirely by the rotary blowers, either class being capable of a reversal of rotation to force air into or exhaust air from the mine, as desired. The power required to drive the force-fan depends upon the volume and pressure of air exhausted or discharged; but the rule usually followed for computing the net power in a given volume at different pressures is to multiply the number of cubic feet delivered per minute by the pressure in pounds per square foot at the blower, and the product by 0.00003; the quotient will give the net horse-power required to drive the fan.

The Centrifugal Fans, which are used almost exclusively in America, may be divided into two great classes: (1) Those which are called open-running, by which we mean that they are free and discharge their air all around the circumference; and (2) those called closed-running fans, which have but a restricted opening for the discharge of the air. Those of either class are made large in diameter and are driven at a relatively small angular velocity, though a few, such as the Schiele, are of small diameter, running at a high angular velocity. They produce large volumes of air at a low pressure, and may be reversed in motion to exhaust or to force air. The diameter of the fans of this class may be and is occasionally as high as 50 feet, those of small diameter being regarded as unnecessarily cumbrous. The action of all fans is based upon the general law that bodies in motion tend to travel in straight lines, resisting any attempt at diversion from this path, in consequence of which, when the fan is set in motion, its blades come in contact with its interior air, the particles of which are at rest and

resist rotation. When, however, the particles do move, their endeavor to travel in straight lines results in their making for the circumference, producing thereby in the central portion of the fan a partial vacuum, which is replaced by the air external to the fan. So long as the rotation of the blades continues, so long will this current be produced and maintained, the pressure of which will increase as the peripheral speed increases.

Open-running Fans.—These fans include the Waddle, Biram, Naysmith, and Hopton patterns, all of which are essentially similar to the first named. The Waddle is a self-contained fan, in that there is no fixed casing, and the whole machine revolves. Its form is practically that of a light hollow disc of wrought iron, the blades and casing being wholly riveted together. The air enters by a straight lead at one side only, and passes through curved and gradually narrowing channels to the circumference, the blades being bent at first to incline slightly backwards, the alternate blades extending not more than one half the distance between the circumference and the inlet. The passages, by their contraction, are so made that the circumference at any point multiplied by the cross-sectional area at that point is a constant quantity. The outer circumference of the fan is bell-mouthed.

A fan of 9 feet diameter circulates 80,000 cu. ft. with a water-gauge of 2 inches. One of 45 feet, driven by an engine with $40'' \times 42''$ cylinder, at a boiler pressure of 80 lbs. per square inch, has given a volume of over 550,000 cu. ft. at 42 revolutions.

The Hopton has an inlet on each side of the central diaphragm with backward-curving blades, and a construction very simple. The revolving portion consists of the arms and blades working between two brick walls.

The open-running fans must, in order to be efficient, discharge their air at a very low velocity, because the energy of bodies in motion increases as the square of the velocity, and that passed by the discharged air is, therefore, so much useless work. It is for this reason that the passages in the more correct open-running fans, like that of the Waddle, are curved

backward. The theoretical depression which can be produced in fans of this type is equal to the height due to its peripheral speed, T, in feet per second.

$$H = T^2 \div 2g = 0.01553T^2.$$

The Closed-running Fans are essentially of a more massive structure than those of the open-running type, being of considerable width as well as of diameter. Of this class of fans the Guibal is a type, the Schiele and the Walker Indestructible being similar in construction. Inside of a fire-proof housing a horizontal shaft is revolved by an engine or dynamo, carrying with it an hexagonal or square frame, on which are built six or eight blades. The blades are flat and slightly curved at their tips, sometimes radially, and often inclined backwards. The clearance between the tips of the blades and the casing is made as little as possible, except for a certain distance at the bottom, through which the air is discharged, the amount of that opening being regulated by an adjustable shutter in a gradually enlarging chimney. The air enters at the centre, whence it passes into one of the intervals between the consecutive blades, which form an *évasée* canal, the speed of exit being less than the speed of entry (Fig. 159).

In the Schiele fan the blades are contracted in width from inlet to outlet, the fan being surrounded by the usual spiral casing, into which the air discharges all around the circumference, the space continually increasing until it reaches the chimney. The blades of the Rateau fan extend to the centre of the fan, and have a peculiar curvature slightly forward, and also a curvature in the line of the fan-shaft. Immediately in front of them is a cone terminating in a point. The Capell fan, of equal power with the Guibal, is smaller, and runs at a higher speed. It has two concentric shells besides its outer casing, in each of which are curved blades with the convex side forward. The air enters the inner shell, is forced out through ports into the second outer shell, where it strikes the concave face of the outer blade, and

thence is discharged at a low velocity through the usual expanding exhaust-flue.

The Champion fan, which is really two fans joined together by a common centre ring, is designed to propel the air with a minimum resistance, the blades having a backward curvature. The use of the inner casing or hood and attendant diaphragm,

FIG. 159.—Section of a Fan.

which are hung on frames, renders it possible to change the current at will, blowing to exhaust, by revolving the hood around the fan without stopping the latter.

The Évasée Chimney.—The depression produced by a covered ventilator with an expanding chimney is twice that of the uncovered or open-running type, and is double the height, due to the tangential speed of its blade-tips. The use of the chimney gives to this type of fan the enormous advantage over the other that the air may be discharged from the fan at a higher velocity without any material loss of energy. The gradually increasing space into which they discharge reduces the velocity and utilizes all the energy in giving motion to the air, while the air is ulti-

mately sent out into the open at a speed so low that practically no resistance is experienced.

The Shutter Regulator.—The volumes which these fans will produce vary directly as the speed of their rotation, and their manometric depression varies as the square of the speed of rotation. Though they may be run at any speed at will, the efficiency of the fan materially decreases when the speed of the tips of the blades exceeds, to a great degree, 5000 feet per minute, or is less than this quantity. The rate, however, which is regarded as normal is 4000 feet of peripheral velocity per minute. Below or above the normal speed a loss of velocity ensues in the discharging air, which alternately is expelled into the chimney or carried with the blades into the fan, there to repeat its circuit. The discharge is frequently followed by a vibration in the fan, to remedy which the sliding shutter (*ab*, Fig. 159) is introduced. Its correct position is only known by experiment in each individual case, determined by the point at which the throbbing ceases with the given speed. Though now universally placed in all fans the "anti-vibration shutter" originated with the Walker fan.

Influence of Shape, Dimension, and Speed of Fan upon its Capacity.—The ratio between the speed and the yield seems to be as the 4th power of the speed to the 5th power of the yield. The relation between the volume resistance of the fan and the power required to drive it depends upon the resistance of the mine which it supplies. These ratios have not been accurately formulated.

Numerous experiments have been conducted upon centrifugal ventilators with the view to determining the influence which the various dimensions of the fan and shapes of its parts will have upon its performance; and the following conclusions are cited from the results of the tests made by R. Van A. Norris, Wilkesbarre, Pa., upon 25 fans, as the influence of: "1st. The diameter on their performance seems *nil;* the only advantage of large fans being in greater width and a lower speed required of the engines. 2d. Width upon efficiency is, as a rule, small. 3d. Shape

of blades shows that the back curvature is better, and diminishes the vibration. 4th. Shape of casing is considerable. The proper shape would be one of such form that the air between each pair of blades would constantly and freely discharge into the space between the fan and casing, the whole being swept to the *évasée* chimney. A large spiral, beginning at or near the point of cut-off, gives in every case a large efficiency. 5th. The shutter on the fan is beneficial. The exit area can be regulated to suit the varying quantity of air, and prevent re-entries. 6th. Speed at which the fan is run. The efficiency is high if the peripheral velocity is large."

Automatic Speed and Pressure Recorders.—Some States require self-recorders on all fans, by which the number of revolutions of the fan shall be registered every hour and such data to be taken and reported. In other States also is required an automatic regulator for the water-gauge. The speed-registers are generally constructed of a metal pedestal erected on blocks at the side of the fan or engine-shaft, a small vertical shaft to which a governor is attached. A small cog-wheel on the lower end geared to a large driver on the fan or engine-shaft communicates the speed to the governor, which, by a system of leverage, raises or lowers the arm to which is attached a pen that presses against a paper dial held in position by a light case of sheet brass. The higher the speed of the fan, the more will the governor raise the lever, and consequently the pen register. The time is recorded by a clock to whose shaft the dial case is attached. In other devices the dial case is a cylinder in which is rolled a sheet of paper turning on a horizontal axis, which is also the continuation of the shaft of the clock. These instruments perform the work expected of them with great satisfaction.

Comparison of Fan and Furnace.—The ventilation current from a fan is affected by a low barometer or a high temperature. Either one requires an increased degree of rarefaction just as from a furnace. Moreover, as the depth of the mine increases, the work devolving upon the fan proportionally increases, because normally the air becomes denser; with every additional

thousand feet of depth, an increased rarefaction or depression of 0.4 inch of water-gauge is necessary. Compared with the furnace its efficiency decreases with the depth of the upcast until, at a certain depth, it becomes an open question between the relative merits and demerits of fan and furnace, as to which will be the more economical. For shallow works the exhaust-fan undoubtedly takes precedence. At the depth of a thousand feet a large furnace will equal a very imperfect fan, consuming 20 lbs. of fuel per hourly horse-power; a good fan and condensing engine will be cheaper than a furnace down to the depth of about 4000 feet. Taking cognizance of the objections to the furnace, it must also be borne in mind that machine ventilators are subject to serious objections, since during the time of their repair the mine must remain unventilated, whereas with a furnace, after its fire has been extinguished, a considerable circulation will still continue in the upcast for some time. Auxiliary ventilating appliances should be supplied against any emergency which arises during the repair of the fan.

The Theory of the Fan.—The theoretical depression of a fan, the difference between the pressure of the entering air and the discharged volume, is represented in terms of a motive column of the density of the inflowing air. An ideal ventilator will produce a depression which is twice the height created by the tangential speed of the tips of the blades. If, then, H be the height of the motive column due to the velocity, T, of the fan-tips in feet per second, H will equal $\frac{T^2}{g}$; but imperfections of detail prevent such an initial depression being attained, and representing them by a coefficient K, which is always less than unity,—reaching 0.85 in Guibal fans, but more often being below 0.6 in the average construction of fans,—the fan approaches an ideally perfect one when K approximates to unity. The yield of the fan then in barometric depression, or its useful effect, is $H=\frac{KT^2}{g}$. Various enfeebling causes reduce the capacity of the fan to determine the value for K. The quantity of air

which passes through an orifice is equal to the product of the area and the velocity when no friction is encountered; but when any fluid flows through an orifice in a plane surface a considerable diminution of the discharge takes place, because of the contraction of the steam resulting from the convergent flow. The coefficient corresponding to this contraction, known as the *vena contracta*, is 0.65; hence with a given velocity, T, and a head, H, under the conditions modified by the coefficient K as above, the discharge of air per second will become

$$q=0.9194a\sqrt{KT^2}.$$

Hence it is evident that if the capacity of the mine is such that it is incapable of delivering to the fan the volume of air equal to the body capacity of the latter at a given speed, the frictional resistances encountered in the mine will reduce the efficiency of the fan by some quantity which is usually comprehended in the symbol a, representing the area of the mine's "equivalent orifice" in square feet. Experiments have demonstrated that when a is 20 square feet, only 65,000 cubic feet of air are obtained per minute for the fan peripheral speed of 5000 feet per minute; but when the mine resistances have been reduced until its "equivalent orifice" is as large as 100 square feet, 280,000 cubic feet of air are obtained from the same speed of fan. Having the value for this friction, which in earlier days was known as the temperament of the mine, we are enabled to understand the conditions under which the ventilator is working and to provide for them.

The Equivalent Orifice of the Fan, which is designated by the symbol o, may be determined in a like manner. It measures or represents the orifice in a thin plate which offers such a resistance to the flow of the current, Q, as is equal in effect to the aggregate resistances encountered within the fan from its imperfections. If H is the theoretical depression which the fan should produce when moving at a tangential speed, T, per second, and h represents the actual or the effective depression which is pro-

FIG. 160.—Details of a Fan House.

duced upon the air as measured by the water-gauge, then $H-h$ is the head wasted by the fan in its construction and may be represented by h_0, which measures the head corresponding to the equivalent orifice. In large fans its value varies from 16 to 80 square feet.

The head lost in the fan, represented by h_0, is equal to $H-h$, the velocity due to which may be determined by the expression

$$v_0=\sqrt{2gh_0}.$$

As the value of h_0 approaches zero and that of h approaches H, the fan approaches an ideally perfect ventilator. The actual velocity through the orifice of entry is $0.65v_0$, whence the area of the orifice o, which equals the quantity flowing per second, divided by the velocity of the flow, has the following value:

$$o=\frac{q}{0.65\sqrt{2gh_0}}=\sqrt{\frac{q^2}{27.87h_0}}.$$

The density of water being 833 times that of air, the ratio between the water-gauge reading and the height of the motive column, H, is 1:833. To convert the water-gauge reading to a height H of air-column of equal weight in feet, the height of the water-gauge, m, in inches is multiplied by 69.4.

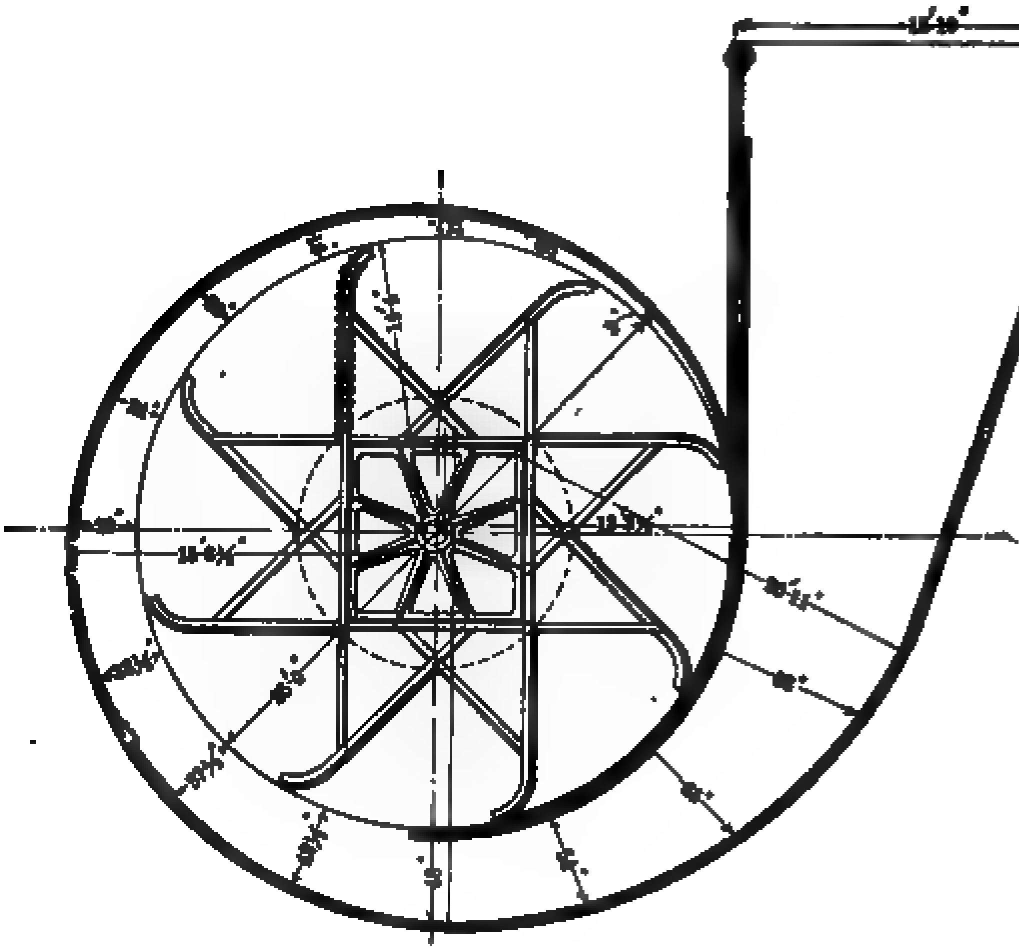
The ratio between the lost head in the ventilator and the effective head represented by the water-gauge is expressed in the equation

$$\frac{h_0}{h}=\frac{a^2}{o^2}, \quad \text{and} \quad h=H-h\frac{a^2}{o^2},$$

whence

$$h=\frac{o^2H}{o^2+a^2}.$$

The Relation between Mine and Fan.—The quantity of flow through a fan, depending on the relations which the area of the mine airways and the condition of their rubbing surface bear to the mechanical condition of the fan, is manifestly depend-



SECTION OF FAN CASING

SECTION ON LINE A-B

PLAN

FIG. 161.—Plan of the Fan House and the Curve of the Fan Housing.

ent upon a proper ratio of a to o, which ratio may be expressed as the "appropriateness of the fan to the mine." When this value is equal to or greater than unity, the fan would be too small for the mine, and it is questionable whether any air would flow under those conditions. As the ratio becomes smaller, the conditions become more favorable for the fan. When approximating a ratio of 0.3 the orifice of discharge of the fan is to be considered as having a fair working ratio. More air is obtained by a given fan and at a given velocity when a is large than when a is small, for, no matter how well constructed the fan may be, it cannot provide a quantity equal to its body capacity unless the mine can pass this quantity. The effective work done upon the air is less in the latter case than in the former for a given volume of air. The mechanical work of centrifugal force is $0.0000340(T^2 - V_2^2)$. In this T is the circumferential velocity and V_2 is the absolute velocity at expulsion, due to compression from centrifugal force. As V_2 increases, so the work on the departing air, and proportionately the effective work, decreases. The use of the funnel-chamber reduces this quantity to $\frac{1}{4}$ or $\frac{1}{5}$, and the work lost to 4 or 5 per cent.

The Efficiency of the Fan.—The manometric efficiency is the ratio between the effective pressure, P, and that due to the centrifugal velocity, while the mechanical efficiency is that of the horse-power in the air to the engine-duty. With fan properly constructed, the efficiency approximates about 6 per cent. In experimentally measuring the efficiency of a fan, it is customary to determine the dynamometric resistance and internal friction when its orifices have been cut off from any communication with the mine, the air being then drawn from the atmosphere and, after passing through a fan, expelled at its throat. Counting the rate of revolution and estimating the volume of air which is moved, the power necessary to overcome this friction is determined and expressed in feet of the air-column whose weight equals the aggregate friction. This quantity, divided by the theoretical head corresponding to the velocity of the fan, determines its efficiency under the conditions named. The fan is

giving its maximum efficiency when "its body capacity just exceeds the quantity the mine will pass at a gauge pressure, F, due to the speed of rotation of the fan."

The Design of a Fan.—In designing a fan for a given mine, the essential elements are Q and m. These given, the diameter, the peripheral speed, and the length and width of blades, as well as the direction of their inclination, must be determined by the engineer. As to diameter, it may be said that the slow-running fans are regarded as cumbersome and costly, requiring expensive foundations. Large fans may be run at a lower rate of revolution and produce the same tangential speed as would a fan of small diameter. Insomuch as speed is the important factor in the construction of ventilators, due consideration must be given to this question, which is determined by local conditions of place, economy, and mechanical simplicity. A convenient rate of revolution for a fan directly connected with the engine is about 60 per minute. The body capacity of the fan should be large enough to maintain the required pressure, P, without great variations in speed. Though the practice of European engineers tends towards the rate of tangential speed which represents 5000 feet per minute and over, in this country 4000 may be considered as normal. In any event, if the calculation and design be made on the assumption of either normal speed, it will be possible, when an emergency arises, to increase the speed sufficiently to give a volume nearly one tenth greater than the normal quantity. Moreover, when the mine is dry and dusty, it will be possible to turn the whole volume of the excess into any or each single split, through which it may be drawn, clearing away fine dust and moisture.

The entry for the air should be made easy and large, preferably divided into two inlets, one on each side, with a diaphragm to prevent the currents from conflicting. This necessitates a wide fan, which, however, gives a volume proportionately greater than what is to be had from a single fan with a single large inlet.

The length of the blades of the fan should be only a little greater radially than the difference between the radii of the

fan and its inlet. With a large inlet the blade necessarily is shortened, and when pressure is desired the blade length should be increased to as large a quantity as possible by providing two inlets. Notwithstanding that the width of the fans is much greater than would be obtained by substitution in the formulæ following, it is certain that the latter dimensions correspond to a greater efficiency. M. G. Hanarte concludes that "the Guibal fan has always been eight or nine times too wide, and the Capell is nearly as bad."

The shape of the blade should be such as would present to the circumference of the outlet an inclination following the resultant of the movement of rotation and of the movement of the air penetrating the spaces between them. The blades of open-running fans curve backward. The backward curvature is conceded to give a freer delivery, and the forward curvature at the tips a higher water-gauge pressure. The number of blades is seemingly a matter of indifference, though the limit may doubtless be determined by the inevitable friction produced by the excessive surface of contact when too numerous. The friction varies as the cube of the section of space between two vanes. As to the shape to be given to the casing, it will be noticed that the original Guibal fan had no spiral, the tips of the blades revolving but two inches clear of the casing, and the spiral enlargement beginning at the angle of about 67° 30′ from the lower vertical radius. Those fans presenting a large spiral beginning at or near the cut-off and increasing about six inches for each 45° up to 275°, and thence widening by an increasing increment to the *évasée* chimney, appear to give larger efficiency by allowing for the slackening of the speed of the air, and discharge the air with less energy at the exit. M. G. Hanarte concludes that the spiral envelope is not necessary.

Below are given formulæ for the computation of dimensions of a Guibal fan in accordance with the data indicated above. All dimensions are in feet.

D = diameter of the fan between the blade-tips: $Q \div 200 = D^2$;
l = length of the blades in feet;

r = their radial length $= 2.61m$;

x = their width $= A \div 2\pi s$;

A = aggregate area of the one or two inlet-ports in square feet (radius of each central inlet, s) $= Q \div 1300$;

N = number of revolutions per minute;

T = peripheral speed of fan per second $= DN \div 19.0985$;

V = theoretical velocity per second due to head H;

v_1 = velocity of the centre of gyration of air-column between the blades $= \pi(D-r)N \div 60$;

ρ = radius of gyration of the mass of air $= \frac{1}{2}(D-r)$.

W = weight of the unit of revolving air-column per foot of fan width $= 0.0766r$;

F = centrifugal force of the air-mass in lbs. per square foot of discharge area or of the housing $= Wv_1^2 \div \pi g = 0.0007578rv_1^2$;

V_2 = velocity of air discharge per second $= \sqrt{\dfrac{(F-P)1,800,000}{2130+P^2}}$;

Z = minimum area of discharge-port $= A \div 2$;

h_0 = fan resistance, measured in feet of head, $= H - h$;

O = area of orifice offering a resistance to the flow of Q cubic feet of air per minute, equal to that of the fan;

q = quantity of air discharged by the fan per second in cubic feet $= 0.65 V_2 Z$;

Q = quantity of air discharged by the fan per minute in cubic feet $= 60q$;

m = mine resistance in inches of water-gauge;

P = mine resistance in pounds per square foot $= 5.184\,m$.

In Figs. 160 to 162 are illustrated the details of the ordinary pattern of fan which is designed in accordance with the conditions indicated above. As, fortunately, neither the Guibal fan nor the shutter is subject to patent, the working drawings here given may aid the construction engineer.

When the conditions are satisfied by the revolution of the fan of proper proportions, the centrifugal pressure of the fan should produce a depression, F, equal to or exceeding P, the mine resistance, in order that the requisite discharge through

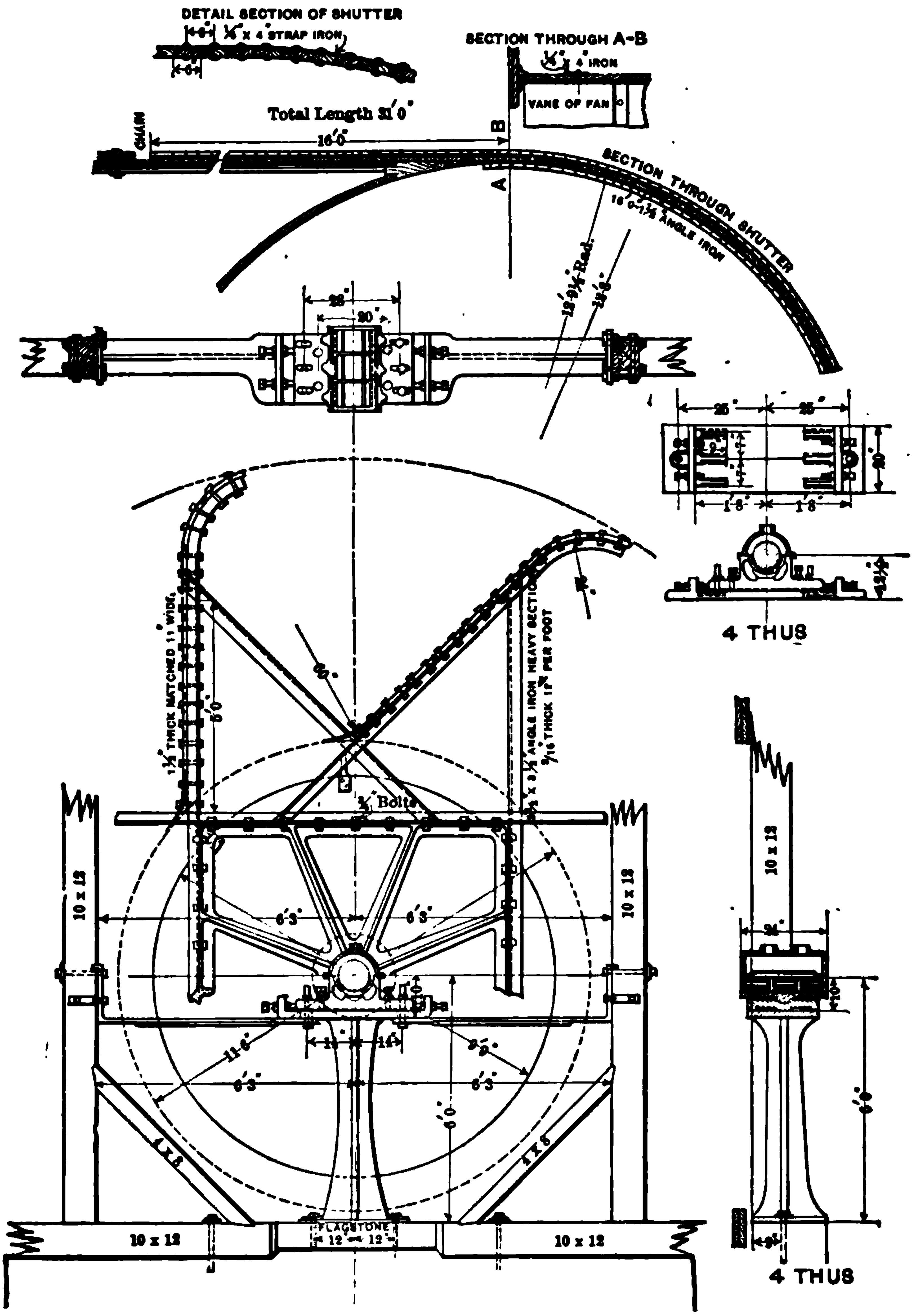


Fig. 162.—Details of the Fan.

the outlet should equal the desired quantity Q. When it is discovered that the volume of discharge is deficient, the fan dimensions D or r should be enlarged or the rate of revolution increased. Below is a brief table indicating the theoretical water-gauge depression in inches for the corresponding peripheral speeds in feet per second.

Speed in Feet of the Periphery per Second. T	Depression in Feet of Air-column. M	Depression in Inches of Water-gauge. m	Speed in Feet of the Periphery per Second. T	Depression in Feet of Air-column. M	Depression in Inches of Water-gauge. m
30	27.95	0.411	75	174.69	2.569
35	38.04	0.559	80	198.75	2.922
40	49.69	0.731	85	224.38	3.299
45	62.82	0.924	90	251.55	3.699
50	77.63	1.141	95	280.28	4.122
55	93.94	1.381	100	310.55	4.567
60	111.80	1.644	105	342.39	5.035
65	131.18	1.929	110	375.77	5.526
70	152.17	2.238	115	410.71	6.039

EXAMPLE.—Required the dimensions of a fan to provide 125,000 cubic feet of air against a mine resistance of 2.5 inches.

At a normal rate of 65 feet per second, the diameter becomes 25 feet; the area of the inlets is 96 square feet, the diameter being 11 feet; the radial length of the blade is 6.5 feet; the minimum width of the blade is to be 2.8 feet.

As $\rho = 9.25$ feet and $v_1 = 51$ feet, the centrifugal pressure, F, becomes 12.7 pounds per square foot of radial column; and the velocity of discharge 31.6 feet per second, which with a minimum area of discharge-port, Z, of 48 square feet, would furnish less than 60,000 cubic feet. The mine resistance exceeds the standard allowance of 1 inch of water gauge for each 100,000 cubic feet of air. The mine airways should be enlarged or the fan operated at a higher speed. An increased rate of 70 revolutions per minute will produce a ventilating pressure of 22 pounds per square foot. The blades may be lengthened and two inlet orifices be provided, each of 48 square feet in area.

At the peripheral velocity, T, of 91.66 feet per second the theoretical head of discharge is 261 feet. But the effective head, h, against which the fan is operating, measured by the water-gauge, is 166.66 feet. Under the conditions of operation, therefore, the loss of head, h_0, in the fan is 94.34 feet; since the equivalent orifice of the mine is 21.2 square feet, the equivalent orifice of the fan, O, is 41.2 feet. The ratio of a to O, nearly one half, represents a fair working ratio of appropriateness of fan to mine.

Electrically driven blowers and ventilating-fans may be connected either by belt, gearing, or mechanical coupling. The compound-wound D. C. machine is preferred for fans on account of its good speed regulation and freedom from racing.

Alternating-current induction-motors of the squirrel-cage type would give equal satisfaction.

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CHAPTER XIV.

DISTRIBUTION OF THE AIR.

The Effective Motive Column.—It has been assumed that the work done upon the air is totally effective in the mine; that, with a given M and P, the calculated quantity of air is obtained without loss; that the momentum, once imparted to the air, would be available to carry it through the mine and out. This is not so. Friction consumes some of the kinetic energy; only a fractional part of the theoretical velocity is realized in practice. The rough sides of the galleries and rooms, their sharp corners, and the diminished areas offer resistances to the passage of the current that consume often 90 per cent of the power. Moreover, the subtle air under pressure seeks to escape at every opportunity, and some portion of the precious fluid is lost into the goaf, through doors and at crossings. A certain mine theoretically required a pressure of but 1.2 lbs. per foot to give rise to its current, yet the friction was such that 11.8 lbs. were actually necessary to create the velocity. Not infrequently the ratio between M (to which the generation of the final velocity at the top is due) and M', the head actually necessary to overcome resistances, is as low as 1 : 18. In other words, only 5.5 per cent of the work done upon the air is usefully expended. Any means of reducing this loss is to be welcomed.

The Frictional Resistance of Air in Mines.—Let us examine into the laws governing the movement of fluids, that we may reduce this friction to a minimum, and obtain salubrity, safety, and economy with the least outlay. The air which enters the mine from the downcast is distributed to the rooms and chambers in proportions varying with their several needs; or the current

as one mass sweeps through the main way, along working faces, thence by return airway over the furnace or to the fan. The resistances encountered depend upon the ratio of the area of the surface rubbed to the area of the conduit, and upon the coefficient of air friction against rock. A satisfactory value for the coefficient has not been obtained: the records of experiments show it to vary, as in water, according to the nature of the conduit and the velocity of the flow. The coefficient varies with the nature of the rubbing-surface, and consequently differs in various air-passages of the same mine; nevertheless, the numerous experimenters have announced values for the coefficient of friction of air in mines for each foot of rubbing-surface and for a velocity of one foot per minute as varying between 0.000000008585 and 0.0000000219, with an accepted average of 0.00000001 lb. This value for f, the coefficient, is measured in the pounds pressure per square foot. Measuring the height of an air-column in decimals of a foot, the value for the coefficient of friction f' has been found to be 0.000000010635 to 0.00000026881, the two extremes of values for f' being heads of air-column corresponding to the values given for dynamic pressures, f. In other words, an area of 1 square foot of rubbing-surface presents to the passage of an air-current moving at 1 foot per minute a frictional resistance of 0.00000001 lb., or an equivalent resistance of 0.000000125 feet of head of air-column.

Let l be the length, m the perimeter, and a the area of the gangway, through which the air is coursing at v feet per minute, and the rubbing friction is found experimentally to be $flmv^2$. Imagine a piston, fitting air-tight in the passage; to just move it against the resistance requires the expenditure of a force pa, in units of pounds and square feet. Therefore the loss of power due to friction is $pa = flmv^2$, and the loss of head in feet, $p = \dfrac{flmv^2}{a}$, or in pounds, $p' = \dfrac{f'lmQ^2}{a^3} = \dfrac{f'lmv^2}{a^3}$.

Shape and Dimensions of Airways.—Cognizance must be taken of these frictional resistances by ascertaining, for various

portions of the mine, the elements involved and comparing the value for p, thus ascertained for a given mine, with that usually considered as the standard of comparisons. The normal mine should be capable of delivering 100,000 cu. ft. of air to the men throughout its working without causing a frictional resistance greater than 5.2 lbs. per square foot in the aggregate. Should it be ascertained that the mine shows a larger water-gauge, it is evident that there is room for improvement.

The lines along which improvement may be had involve an increase in the area of the cross-section, or a change of shape, such as to give a greater flow with less frictional rubbing-surface. The conditions under which the airways are driven prevent any reduction in the coefficient of friction. The friction increases with the square of the velocity, and in order to increase the air-current the velocity may be increased, but this increases the frictional resistances rapidly. Assuming the value for f of 0.0000001, then the quantity that will flow through 1 sq. ft. of area of passage at different velocities, as stated below, in a length of one mile, will be as indicated, and with corresponding resistance, p, as follows:

	Feet.		Cubic Feet.		Lbs. per Square Foot.
Velocity	1,	quantity	$Q = 60$;	then	$p = 0.76$
"	2,	"	$Q = 120$;	"	$p = 3.06$
"	5,	"	$Q = 300$;	"	$p = 19.00$
"	10,	"	$Q = 600$;	"	$p = 76.03$

The frictional loss is directly proportional to the perimeter, and inversely as the area. Thus, a local contraction of the air-passage by the use of the partly open door, or the accidental accumulation of a pile of waste, or gangue, will materially diminish the volume of air passing through it and increase in a large measure the resistance to such flow as does occur. It is therefore desirable to select such a shape for the airway as will make it as spacious as circumstances will permit consistent with the dimension of the rubbing-surface. The circular form most nearly meets these requirements, but we are restricted, as a rule, to the

rectangular or the trapezoidal cross-section. Aside from the theoretical consideration, the form of airway is modified by practical conditions. In American mines the circular airway is rarely adopted, and only in very rare instances is the Stanley header used as in England. There a circular lining follows the machine very closely, and the proper form can be retained in the finished airway. The expense of making this form prevents its common use in America. The more usual form is one having a semicircular arch roof and smooth walls as being the nearest approach to the ideal cross-section.

As showing the comparisons between the resistance afforded by a square, trapezoidal road and the round roadway of various dimensions and velocities, as stated below, the following table will be of service. It shows the pressure in pounds per square foot due to the flow of air through one mile of airway with the coefficient of 0.0000001 lb. per square foot.

Dimensions of Airway.	Diameter of Airway.	At Velocity of 1 Foot per Second.		At Velocity of 5 Feet per Second.		At Velocity of 10 Feet per Second.	
		p	Q	p	Q	p	Q
Feet.	Feet.	Pounds.		Pounds.		Pounds.	
1× 1	1	.7603	60	19.00	300	76.03	600
2× 2	2	.3801	240	9.50	1,200	38.01	2,400
3× 3	3	.2534	540	6.33	2,700	25.34	5,400
4× 4	4	.1900	960	4.75	4,800	19.00	9,600
5× 5	5	.1520	1500	3.80	7,500	15.20	15,000
6× 6	6	.1267	2160	3.16	10,850	12.67	21,600
7× 7	7	.1086	2940	2.71	14,700	10.86	29,400
8× 8	8	.0954	3840	2.38	19,200	5.94	38,400
9× 9	9	.0844	4860	2.11	24,300	8.44	48,600
10×10	10	.0760	6000	1.90	30,000	7.60	60,000

It will be noted then that two galleries having a cross-sectional area of 5′×5′ would require twice as much power to carry the same amount of air as would a single gallery of 10′×10′. The pressure necessary to drive 25 cu. ft. of air per second through one of the small galleries is 3.8 lbs. over the total area of the road. The pressure necessary to drive 100 cu. ft. per second through a large gallery would require the pressure of 7.6 lbs. over

the total cross-sectional area. This is equal in amount to resistances of both small galleries, which, however, are transmitting but 50 cu. ft. of air per second.

The frictional resistance varies with the length of the gallery. The volumes of air passing through two having the same resistance will be inversely as the square root of their lengths. For example, a gallery 1600 feet long carrying 6000 cu. ft. of air offers the same resistance and consumes the same amount of power as one of equal area which is 711 feet long delivering 9000 cu. ft. per minute.

The following table shows the length of roads offering a resistance equal to 1 inch of water-gauge under the conditions given:

Square Airway.			Round Airway.			Length of Road in Yards.
Dimensions of Airway.	Area.	Periphery, *m*.	Diameter of Airway.	Area.	Circumference, *m*.	
Feet.	Sq. Feet		Feet.			
1× 1	1	4	1	.7854	3.1416	481
2× 2	4	8	2	3.141	6.2832	962
3× 3	9	12	3	7.068	9.4248	1444
4× 4	16	16	4	12.566	12.5664	1926
5× 5	25	20	5	19.635	15.7080	2407
6× 6	36	24	6	28.274	18.8496	2888
7× 7	49	28	7	38.484	21.9912	3370
8× 8	64	32	8	50.265	25.1328	3850
9× 9	81	36	9	63.617	28.2744	4333
10× 10	100	40	10	78.540	31.4160	4814

In using this table it will be remembered that the volume of air and the lengths of the road are directly proportional to the frictional resistance, p, the other conditions of area and periphery being the same. For example, a gallery which is 1200 yards long, carrying 6000 cu. ft., would offer one half the resistance that a road 2400 yards long would, or, in other words, if the cross-sectional area be $5' \times 5'$, the water-gauge would read ½ inch in the short gallery and 1 inch in the long one. Again, the roadway of $7' \times 7'$, which is 6740 feet long, would offer a resistance measured by a water-gauge of two thirds of an inch.

Calculation of Mine Resistance. — It has already been remarked that the water-gauge measures the drag of the air in a mine, and thus serves to indicate the pressure and head corresponding to the motive column, M. The pressure varies from $\frac{3}{4}$ inch for easy to 4 inches for difficult ventilation (from 3.9 to 20.7 lbs. per square foot). In anthracite mines it is about 2 inches. The motive column, which is to just maintain this pressure against resistances, should also be sufficient to create a final or exit velocity in the shaft. If the entire current traverses the mine unbroken, the resistance in the shaft or entry is only a fractional part of the mine friction indicated by the water-gauge, and the following formulæ apply with sufficient accuracy:

$$V^2 = \frac{pa}{flm}, \qquad Q^2 = \frac{a^3 p}{flm}.$$

The value for f is to be taken always in the same unit as that for p. In other words, if the mine resistance, p, be given in pounds per square foot, the corresponding value for f is taken as equal to 0.0000000219; or if the value for p' is given in feet of head of motive column, M, the value for f' is then 0.000000269.

If the airways in the mine, the resistances of which are to be calculated with a view to determining the necessary ventilator pressure to produce circulation, are all of the same dimensions, the calculation of the lost pressure may be made in one operation by proper substitution for the length, periphery, and area of airway and the velocity or quantity concerned. The value of the frictional resistance, p, thus engendered in the mine corresponding to the water-gauge height, m, and of the velocity of the air-current, added to that of the pressure, p'', requisite for the generation of the velocity, determines the motor pressure required. Often p'' is very small compared to p, and may be even neglected without sensible error; but when it is large, the actual ventilating pressure, which must be supplied by the force-fan, or the manometric depression to be produced by a furnace, or exhaust-fan, must be such as exceeds the sum of $p + p''$.

When the airways of the mine are of various cross-sections, the resistance offered by them in the aggregate must be deter-

mined by adding together the separate values for p, calculated for each differing cross-sectional area and length. When the air-current is "split" into several smaller branches, and circulated through an equal number of divisions of the mine, more or less equal in length, with volumes of greater or less velocity, the value for p must be calculated in each division or district separately; and for each differing airway the aggregate resistance in each division is the sum of the resistances encountered in each of its various galleries. The sum of the several frictional losses of head or of pressure, and that pressure or head which produces the final velocity at the mouth of the mine, is again equal to the ventilating pressure demanded of the motor.

Formerly it was the practice to meander the air through all the galleries of each lift before expelling it (Fig. 163). This involved heavy pressures, enormous airways, or a velocity dangerously fast, and the last gang, fed by the departing current, would receive an irrespirable atmosphere, vitiated by the emanations from all previous sources. There was nothing to commend this pernicious system, and it is certainly a matter of congratulation that it has become obsolete.

Multiple Air-circuits.—Mr. J. Buddle introduced a system of ventilation for fiery mines that has everything in its favor. This system was known at first as "coursing the air," and now is termed "splitting the air," the inception of which is due to Carlisle Spedding or his son, of Whitehaven, who introduced it in 1763. By it the aggregate quantity of air is increased, the dangers of explosion are lessened by confining its train of evils to one portion of the mine, and power for ventilation and haulage is saved, since it goes hand in hand with the method of panel-working (Fig. 14). Each panel of the mine is completely isolated from the contiguous districts by barrier pillars, and is ventilated separately by delivery to it of a portion of the volume of the intake which does service in that panel, to be afterwards discharged into the return airway, where it rejoins the exhaust from the other districts. The electric distribution for purposes of illumination and the water-supply of a town are conducted on identically the same prin-

ciple, i.e., that which recognizes the tendency of a fluid to seek a shorter and easier escape from confinement. With a number of conduits receiving at a common point a volume of fluid from a larger conductor, each will convey a fractional amount of that original bulk which is inversely proportional to the resistance offered by its entire rubbing-surface. If the several conduits again meet to discharge their individual volumes of fluid at a common point into a common reservoir, the pressure at the point of discharge is the same at the mouth of each and every pipe. Likewise the pressure at the point of union is the same in the ends of each and every pipe. The loss of head or of pressure due to the flow of the given quantity of fluid through each conduit is then the same. If the original bulk is allowed naturally to subdivide, the amount of fluid in the several branches will vary in an inverse ratio with the cross-sectional area of their conduits. This is equally true of the circulation of air through mine-galleries or districts, of water through branching pipes, or of electricity through connecting wires in the circuit.

The Process of Balancing the Resistances.—In planning the ventilating system for a mine which is to be divided into a number of districts for ventilation purposes, the practice is to calculate for each separate district its aggregate resistance to the flow of the volume required for a known number of men employed there, and for a dilution of the gases evolved in that district. Several separate values for p are thus obtained. But these district resistances must be equalized or else the inlet-current will be so subdivided at the point of distribution that the large bulk of the air will pass through that district which offers the least resistance; while to that district offering the greater resistance the volume there circulating will be small. This is usually the reverse of the requirements; for, generally, that district offering the smaller resistance is the shorter one, having less men in its circuit, and therefore requiring a smaller volume of air; while that district presenting the greater resistance to the flow of the current is either more extensive, has a greater volume of goaves, or contains more working places, and hence demands a very

large fractional part of the main current. In order, then, to automatically deliver to that district requiring more air, which at the same time offers a greater resistance, the area of the conduit throughout its course or the area of the orifice at the central point of distribution must be made sufficiently large to tempt through it, or into it, the requisite amount of air, leaving to the smaller district, which requires less air, an orifice of entry which is comparatively small. By so doing the differences in pressure for each and every district, between the point at which the splits of the fresh-air current are made and the point at which the return currents from the same splits reunite, can be equalized to that of the one offering the greatest resistance. Then the current will naturally divide according to the areas of the inlets or of the passages, and each district will receive its apportioned fraction of the incoming air. Hence, whenever the ventilation of the mine is to be split into several currents and the air is to be apportioned in accordance with the demand, the mine foreman, having calculated the relative values for the head lost in each, determines by proportion, as will be seen, the comparative area of inlet to be provided the several districts at the point of distribution, and there, by means of doors and other regulators, furnish to the given district the area desired.

The measurements of the water-gauge pressure, or loss in head between the beginning and the end of the split and the velocity of the flow of air, are made in the intake; and while it is not always possible to subdivide the current at a common point of distribution, this should be done as near to the downcast as circumstances will admit. The same may be said of the point of reunion. Otherwise the resistance of the intermediate ways and of the entries must be determined and provided for, as may be seen in the example given below. The aggregate resistance of the intermediate ways of the several districts through which the air is circulated determines the maximum number of splits which are possible.

Simple as is the theory, and satisfactory and economical as is the plan when well developed, it is not easy of execution.

The success of the plan involves an exact manipulation and great skill in taking due precautions to balance the various factors, to determine the equilibrium designed, and to prevent one panel or district from receiving too brisk a current at the expense of others. Hence, while it is eminently desirable to apply this theoretical distribution, its difficulty is recognized, and it has become the practice of the foreman to approximate the desired conditions by making repeated tests upon the quantity of air flowing in a given circuit, which, if insufficient, is provided for by enlarging the inlet area for the given district and watching its reaction upon the volumes in the other dependent splits. In shallow workings, though the mine may be extensive, the practice is an inexact one in many cases. It may often be cheaper to sink a new shaft to furnish separate ventilation to a district than it would be to undertake to furnish an elaborate system of splits.

The Power Required for Ventilation.—Though it may not require a demonstration to show that the subdivided splits of the current are productive of greater economy in ventilating power, attention will be called to the fact that the ventilating force in horse-power, necessary to deliver a volume, Q, against a mine resistance, p, is measured by the expression

$$\text{H.P.} = Qp \div 33{,}000 = Qm \div 6365.7.$$

The resistance which would be offered by the aggregate of all the districts to the flow of the entire volume, Q, through the whole length of the circuit is measured by p. Each fractional volume, q, q', q'', etc., passing through only one branch of the circuit would offer a resistance r, r', r'', etc., which is very small compared with p. When the mine boss has adjusted the regulator doors at the point of distribution by altering the respective areas of inlets, the resistances in all of the several circuits are equalized, the work performed in each split is qr, $q'r$, $q''r$, etc., and the aggregate ventilating power is their sum. As

$$q + q' + q'' + \ldots = Q \quad \text{and} \quad r < p,$$

the power required for the ventilation in branches will be less than that for a single current, Q, through the same passages.

The power required for 16,200 cubic feet of air flowing in one column would be capable of producing 70,884 cubic feet of air in five splits, 94,850 cubic feet in ten splits, and nearly 100,000 cubic feet in fifteen splits.

EXAMPLES.—1. A colliery is ventilated by a Guibal fan of 21 feet 3 inches diameter, making 40 revolutions per minute. How many cubic feet will it produce? The air must pass through a main airway 300 feet long, $6' \times 12'$, before being split up into three separate airways, one being 12,000 feet long, $6' \times 5'$; another, 11,c00 feet, of area $6' \times 7'$; while the third is 10,000 feet long and $5' \times 8'$ in section. Required also the water-gauge pressure, assuming the two shafts together to consume 0.226 lb. per square foot in riction.

16,380 cubic feet and 0.9 inch.

The splits are all drawn from a common point of junction.

Theoretically, the fan produces a water-gauge pressure of 0.902 inch (4.677 lbs.). Then the entire mine offers a resistance of 3.504 lbs. (0.676 inch). The resistance of the main airway is $p=0.0000000006279Q^2$. Q, the quantity of air delivered, is divided up into three separate sections, according to their resistances. As p is the same for each, the quantities q may be known in terms of p to be

$$2171\sqrt{p}, \quad 3455\sqrt{p}, \quad \text{and} \quad 3368\sqrt{p}.$$

Now the resistance of the entire mine is equal to 3.504 lbs., plus that of the main airway, plus that of the split, p. From this we know $p=3.504 - 0.0000000006279(8994\sqrt{p})^2$, whence p becomes 3.333 and the r sistance of the main airway 0.170 lb. Q then becomes 16,380 cubic feet, and the quantities received by the three splits are 3940, 6270, and 6110 cubic feet. (The difference in results arises from failing to carry out the decimals beyond two places.)

2. An airway 3000 feet long, $8' \times 4'$ area, is carrying 20,000 cubic feet. How many cubic feet would be delivered if the air was split into three currents, the *power* remaining the same? The sections are 3000 feet long and $8' \times 4'$ area; 3600 feet and $5' \times 9'$; and 4800 feet $6' \times 10'$.

51,736 cubic feet and 11.56 H.P.

The calculated power necessary to drive the quantity of air, Q, through the three sections is equal to the sum of the three powers, pav, of each section.

The benefits that may be derived from splitting the air-current are manifest by inspection of the following case:

EXAMPLES.—1. A mine has two slope entries, $9' \times 14'$ in cross-section and 100 feet long, and such internal resistances as would be equivalent to 8000 feet of a typical airway $5' \times 10'$ in cross-section. What pressure and power will be requisite to propel 16,200 cubic feet?

$$p = 0.2619 \text{ and } u = 4243 \text{ ft.-lbs., for the two entries, and}$$

$$p = 11.19 \text{ and } u = 181{,}383 \text{ for the total.}$$

2. Required the quantities of air that will circulate where there are 2, 3, 5, 10, and 15 equal splits, the pressure remaining the same as above.

After calculating the pressure p for the one current as above, then proceed to ascertain the pressure p' necessary to circulate 16,200 cubic feet in the several cases. These will be found 1.369, 0.405, 0.0874, 0.0109, and 0.0032 lb. per square foot, respectively; as the area of each airway is 50 sq. ft., the aggregate for the equal splits are 100, 150, 250, 500, and 750 square feet in the several cases, while the rubbing-surfaces, lm, are the same (240,000 square feet). The pressures are then apportioned directly to 10,935 lbs., the mine friction of one current as above. Whence, the pressures being as the squares of the volumes circulating, we obtain 33,409 cubic feet, 66,354 cubic feet, 91,692 cubic feet, 103,755 cubic feet, and 105,255 cubic feet, and 373,887 ft.-lbs., 742,504 ft.-lbs., 1,026,030 ft.-lbs., 1,161,010 ft.-lbs., and 1,177,760 ft.-lbs. as the respective powers u.

3. If it be desired to know what quantities will circulate with the same power u, as in Ex. 2, then we have but to apportion the volumes to the cube roots of the powers, u.

Thus $\sqrt[3]{373{,}887} : 33{,}409 :: \sqrt[3]{181{,}383} : 26{,}252$, the volume with 2 splits.
$\sqrt[3]{742{,}504} : 66{,}354 :: \sqrt[3]{181{,}383} : 41{,}479$, " " " 3 "
$\sqrt[3]{1{,}026{,}030} : 91{,}692 :: \sqrt[3]{181{,}383} : 51{,}461$, " " " 5 "
$\sqrt[3]{1{,}161{,}010} : 103{,}755 :: \sqrt[3]{181{,}383} : 55{,}881$, " " " 10 "
$\sqrt[3]{1{,}177{,}760} : 105{,}255 :: \sqrt[3]{181{,}383} : 56{,}419$, " " " 15 "

4. When, however, the splits are not taken from a common point of juncture, the procedure for ascertaining the mine resistances, and, subsequently, for balancing the delivery of air to the several sections, is not so simple. The plan consists in determining the several resistances and the power necessary to overcome them. These are then added as follows: In Fig. 163 D is the downcast shaft, 846 feet deep, $8' \times 10'$ in cross-section, delivering 56,000 cubic feet per minute; by force-fan, 27,750 cubic feet go to the left gangway, while the right gangway passes 28.250 cubic feet of air. The water-gauge stands at $\frac{7}{8}$ inch (4.525 lbs.). Required the volumes of air received by the splits A, B, C, and D.

The distances along the gangway at which the splits are taken are, a, 460 feet from d; b, 960 feet; and c, 1360 feet from d. Dimensions of gangway 6'×12'.

The splits are E, receiving 5000 cubic feet through 100 feet of 6'×12' gangway; 60 feet of 4'×2' break-through; and 60 feet of return airway

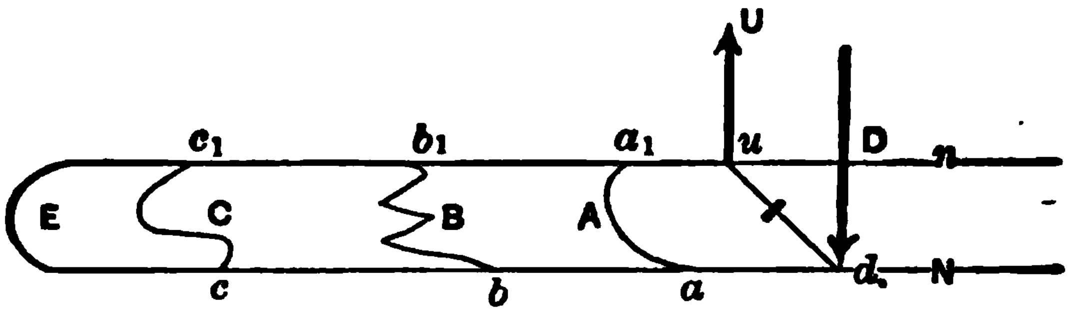

FIG. 163.—Diagram of a Split Air-current.

5'×14'; C, having a resistance equivalent to 2401 feet of typical roadway 15'×6.87', connecting with the return-way at c_1, 941 feet from the upcast U; B, which has 220 feet of gateway 4'×7', 1400 feet of entry 4'×8', and 1000 feet of room 4'×12', delivering at b_1, a point 581 feet from U; and A, which has a resistance equivalent to 1810 feet of typical roadway 5'×10', connecting with the return airway 313 feet from U.

The upcast shaft is 870 feet deep and 12 feet in diameter. It was found that the velocity of the outgoing air was nearly 708 feet. This would correspond to about 80,000 cubic feet of air.

In the return airway, along ua_1, the volume was found to be 41,610 feet. This shows an increase in volume of 13,870 cubic feet, which may be accounted for by the higher temperature of the outtake air or increments to the original volume from gas.

Beginning at E, where 5000 cubic feet are circulating, we find the total resistance to b such as requires a pressure of 0.096 lb. per square foot at c, to overcome it. Hence the same pressure must exist at the mouth of C, where $pa=0.00000021967q^2$. Equating the two values for p (q being unknown for C, and 5000 cubic feet for E), we find that the volume of air of 6410 cubic feet offers a proportionate resistance to that in E.

At the point b the total resistances of E, C, and their gangways must be proportional to those of B, as the relative volumes to be circulated. At b the 11,410 cubic feet passing out produces a pressure of 0.57 lb. per square foot, at which rate, by equating this with the sum of the resistances of the split B, we find that q becomes 3720 cubic feet.

In like manner 12,580 cubic feet will be found capable of passing through A, the frictional resistance at a being 1.19 lbs. per square foot.

The total resistances pa of the various sections are: da, 53.33 lbs.; A, 74.59 lbs.; ab, 17.29 lbs.; B, 16.04 lbs.; bc, 7.83 lbs.; C, 9 lbs.; E, 6.92 lbs.; c_1b_1, 17.67 lbs.; b_1a_1, 23.29 lbs.; a_1u, 91.2 lbs.; D, 323.85 lbs., and U, 356.2 lbs.

The volumes of air are, respectively, 27,740 cubic feet; 12,580 cubic feet; 15,150 cubic feet; 3720 cubic feet; 11,410 cubic feet; 5910 cubic feet; 5000 cubic feet; 17,115 cubic feet; 22,725 cubic feet; 41,610 cubic feet; 80,000 cubic feet; and 56,000 cubic feet.

The pressure mentioned is only that necessary to overcome the rubbing friction of the moving air against the sides. No cognizance has been taken of the pressure requisite to air the mine. That power will depend upon the velocity, and is equal to about 0.17 lb. pressure per foot.

The power requisite to force the air into and out of the mine (11.9 lbs. per square foot) is 666,400 ft.-lbs.

5. An airway $5' \times 8'$ across, 6000 feet long, is followed by a through $2' \times 5'$ in cross-section 300 feet long. To drive 10,000 cubic feet of air through them requires 5.5 horse-power. Their frictional resistances are equivalent to 5.2893 lbs. and 9.114 lbs. per square foot, respectively; the total resistances, *pa*, being 211.57 and 91.14 lbs. A pressure of 302.71 lbs. is therefore requisite at the inlet to overcome frictions of the passage. $302.71 \div 40 \times 10{,}000 = 75{,}667$ ft.-lbs. to propel the air through the first section, in addition to which it requires $9.114 \times 10{,}000$ lbs. to overcome the resistance of the second.

When the "through" precedes the airway, we have no differ nce in frictional resistances, but the total pressure in the first section must be such as to overcome the total resistance of the two, hence it is 302.71 lbs. (30.27 lbs. per square foot). Then $30.27 \times 10{,}000 + 5.289 \times 10{,}000 = 355.601$ ft.-lbs. $= 10.7$ horse-power.

If the "through" is midway along the 6000-foot airway and has 300 feet of the $5' \times 8'$ roadway on each side, we still have the same resistances, yet the power requisite for propulsion is 299,047 ft.-lbs. For each end the friction is 2.645 lbs. per square foot, and for the middle it is still 9.114 lbs. per square foot. The total pressure of the first is 105.784 lbs.; of the second, 91.14 lbs.; and of the last, 105.784 lbs. The pressure at the inlet is 302.71 lbs. (7.5677 lbs. per square foot); at the entrance to the second, 196.924 lbs. (19.6924 lbs. per square foot); and at the entrance to the last, 105.784 lbs. (2.645 lbs. per square foot).

Goaves.—These are the most dangerous places. Here the waste rock, broken coal, slate, dust, pyrites, etc., accumulate in the presence of water which, with the stagnant air, induce a combustion from which sulphuretted hydrogen and other gaseous products exude. It is estimated that the air-space in a goaf is one sixth of the volume of coal extracted, and in it most likely will breed a great deal of gas, of which the sweating of the roof is an infallible sign. Often a water-gauge placed in a goaf-stopping

will indicate by the difference of level in its arms whether or not any accumulation of gas exists behind the stopping. Spontaneous combustion once begun therein, nothing will stop it. For this reason, though aeration is possible, fiery coals should only be worked by a method involving complete removal of the coal, or its replacement by clean waste.

The Velocity of the Air-current.—Spacious air-drifts will then reduce the resistances and also the velocity of flow. It is not desirable that the velocity exceed 500 feet per minute, nor is it comfortable or safe. A speed exceeding 500 feet is equally injurious with stagnation. Several mining commissions of Europe have experimentally determined that, aside from the chilling effect of walking against so rapid and cool a breeze, lamps are not safe: a rapid current incites explosion by driving the gases through, or the flame against, the screens. Many lamps can resist higher-speed currents, but none are safe in over 900 feet per minute. In the old country the air-currents at different parts of the mines vary in velocity—at the coal face often as fast as 900 feet; but here our Davy, Stephenson, or Clanny lamps require protection in such a velocity, which exceeds that of American practice.

The Anemometer.—A miner can approximately estimate the speed of the current by knowing the rate at which he must walk to keep the flame erect, or by noting the time elapsing between the discharge of a volatile fluid or smoke and the time of its arrival at a point a known distance beyond. The anemometer is, however, much the simpler instrument for measuring the velocity of the flowing air. In a case is a series of vanes which are moved by the current, and these by proper gearing turn indices over their respective dials at such a rate that the velocity may be at once read. It does not give accurate results on account of the friction of its mechanism. Each instrument has its own factor, which is not even constant. Biram's and Costello's patterns (Figs. 164 and 165) are most used in America. Their factors are ascertained by occasional test, the anemom-

eter being revolved on a whirling table, and its reading compared with the actual velocity of revolution.

The point selected for observing the velocity should be in a straight gallery, whose sides and roof are a fair average in rough-

FIG. 164. FIG. 165.

Anemometers.

ness, and where there is neither a sudden bulge nor a contraction. The average of several one-minute readings is taken at the place of measurement, near the roof, sides, and centrally. Then the cross-section of the conduit at the observing station is taken.

The Efficiency of a Ventilating System.—One object in determining the velocity of the air circulating in the passage is to ascertain the efficiency of the motors supplying the current. For this purpose the cross-sectional area of the airway at the observing station is measured, and the product of this with the velocity and the ventilating pressure at the given point determines the actual work of the effective power in the air at that point. The power U to move the air is $0.0000303vaP$. The ratio between the result of these observations and the simultaneous indicator diagrams of the engine gives the mechanical efficiency of the ventilating system. The ultimate comparisons of this work in terms of consumption of coal should show for a good system not more than 11 lbs. of fuel per useful horse-power

produced by the fan and 40 lbs. to 70 lbs. per effective horsepower of the furnace.

The Regulation of the Multiple Circuits.—The air entering the mine is conducted to the men by a route as direct as the plan of the workings admits. To those who are at work in the shaft and at the breast of the exploration entries, the current is led by a partition in the shaft or drift-forming conduits for an intake and return current; to the men engaged in extracting coal, the current passes from the main entry to the heading, thence through cross-headings, or branch headings, to the rooms. The air-current passing down the entries by either shaft or slope is split at each level or lift to a right and a left branch. In each lift the current flows to the furthermost point, along the main heading, thence rising into the most remote room, along its face, thence through "dog-holes," progressing toward the hoistway till each room has been traversed, when the current is led to the return or back heading which communicates to the upcast way, there to be joined by the return current from the opposite side of the lift, and finally to unite with the return current rising from the lower lifts which have been fed in like manner.

The main heading is usually the haulage-way, the back heading serving only for the vitiated air. It may be used for a travelling way. The jaw of each room, except that one which is most advanced, is stopped up, which latter, in time, is closed as another room is turned from the heading. As each room must be in free communication with the haulage-way for the delivery of its coal, the stopping there provided must be capable of opening for the passage of the coal. The swinging doors, therefore, which are placed in the entry are always closed to prevent the air from returning at once down the back heading. Here doors are built to deflect it into the rooms, with also a brattice in each room, to direct the current it receives from its neighbor. From the last room, whether working or abandoned, the air passes to the return current. In the lower portions of the room not in the direct sweep of the air-current the air is prevented from

becoming stagnant by the leakage from the main current and the addition of such volume as is swept in or out by the travel of the cars from and to the haulage-way.

The distribution in the long wall is perfectly similar, in which case the gob roads are stopped up by doors to convey the current to the extremities of the workings, whence the air flows along the long-wall face.

In flat seams, the double entries of which have a low grade for haulage from the surface, the similar split system is employed, the splits being taken at the point of union with the main headings. When, by reason of the low pitch, three entries are driven from daylight, the common practice consists in making the central entry the return airway, the air being led into the mine in two currents on either side. This simplifies the subdivision of the currents, which may be effected with few doors. Each district on either side will have but one crossing of its return current with the intake current.

Structural Appliances Required for Controlling the Air-current.—Their object is to confine and direct the air-currents in a given path. They may be permanent or temporary in character; they may also serve as stoppings to prevent any circulation through a given way, or as a means of deflecting the current into some other channel.

In the split system of ventilation several small currents radiating from one given centre of distribution, or branching from a main airway, are returned, after having done service, to another airway, whence they are carried to the surface. Provision must be made for carrying the return current over or under the intake, where they intersect for regulating the amount of air delivered to each branch of the mine; and for flexible stoppings along the gangway, admitting of the passage of trains of cars.

The Means for Deflecting the Current.—These are of seven varieties:

Air-bridges, constructed to carry the return current over or under the airway.

Stoppings, or walls, closing the "throughs" when their service as connectors has been performed.

Partitions of wood or canvas to temporarily divide each airway into two compartments for an inlet to the face and an outlet to the cross-heading nearest to the face, which in turn connects with the return air, back heading, or adjoining room.

Brattices of canvas hung from the roof in the common travelling-way to serve as a light temporary resistance to the flow of air, without impeding the access of the men into the rooms.

Swinging doors in the gob roads and at the jaws of the rooms to prevent the entry of air from the main heading, but to allow admission to the men.

Swinging doors at the return heading at appropriate places to check the immediate return of the current and to deflect it to a room.

Sliding doors placed in the branch airways and fixed so as to offer an area for the inlets to several branches, including the continuation of the mainway.

The Stoppings.—These consist of walls strongly built to resist the pressure of the roof and floor and also to prevent the leakage of air. They are of brick, with their faces set a little back from the wall of the gallery in order to provide for any peeling of the coal from the side of the airways. These walls are built on either side of the airway two or three bricks thick, and often filled with rubble to make a firm pillar and support. Occasionally their exposed faces are plastered with coal-tar or cement to make them impervious They are built about the goaves and other gaseous districts.

The temporary stoppings are of wood, or canvas sheets, more or less firmly framed, and may be movable, as are doors, or fixed, as in the case with curtains and panels.

Air-crossings or Bridges.—These are placed at the intersection of each pair of cross-entries to allow the pure air from the intake to enter the mine without interference from the vitiated air in its exit. These bridges are carried over the roadways and

dispense with the use of doors. They are permanent structures whose position is determined by the plan of the mine.

Overcast.—There are two classes of air-bridges, one in which the return airway is carried *over* the main haulage-way and intake, and the other in which it may pass *below* the intake. Though there are many conditions favoring the undercast crossing the air-bridge prevails (Fig. 166). In this case, at some distance before the point of intersection of the two airways is reached, an excavation is made in the roof sufficiently high and wide to afford an airway of ample dimensions, which excavation is carried above the intake airway the desired distance and down on the other side to the main level of the coal-seam at some distance beyond. The coal close to the sides of the intake airway is left undisturbed. A bridge is then built consisting of a heavy flooring of timber laid above the opening to make a roof for the intake and a floor for the intake airway. It would be cheaper to remove all the coal on the sides and erect stopping-walls to furnish support for the bridge overhead, but these are not always air-tight. It is absolutely essential that the partitions between the return and the intake airway should be air-tight. The cost of an over-

FIG. 166.—A Rock-tunnel Air-crossing.

cast is about $125.00, with a month's labor of common help and ten days of two masons. There would be required perhaps 100 bushels of mortar, 5000 bricks, and 6 cu. yds. of sand.

Many engineers prefer to drive the overcast through the solid rock to insure a tight airway and one which is not liable

to be influenced or affected by the accidents in the intake. These are more expensive to build, but have many advantages.

The Undercast.—This air-crossing is built below the floor (Fig. 167) of requisite dimensions and far enough beyond the intake on

FIG. 167.—An Undercast Air-crossing.

either side to leave a secure and undisturbed pillar for a stopping in the latter. It is framed and boxed as is the overcast. In a few cases this construction may be cheaper than the overcast, and is preferred when the roof cannot well be disturbed. But the liability to undermine the coal-pillars which rest upon the floor of the seam and the risk of creep constitute a serious objection to it. The depression which is made in the floor furnishes opportunity for lodgement of water and mine-gas and may become in time absolutely impassable. It is always subject to constant flooding.

How Should Air-bridges be Made?—It is a disputed question whether air-bridges should be carried through the solid rock or be built in the roof. Both serve equally well their purposes under general conditions. But the effect of an explosion upon the two is different. The rock terminal airway remains intact and the ventilation of the mine is unimpaired. The boxed bridge-crossing is usually destroyed by the force of the blast, and with it the air-current. Of course the force of the blast is confined to the airways, and its violence may produce heavy falls which will afterwards present obstacles to the rescuing party, when the work of rehabilitation begins, though the number

of such violent explosions that will shatter the roof will be small.

The timber overcast bridge furnishes a safety-valve for the force of the explosion and prevents much damage. In such event the reestablishment of the air-current after an explosion becomes a difficult matter. The wooden partitions may be destroyed by a moderate explosion, and the number of occasions when the rescuing party must build bridges and stoppings is larger than those in which it must reopen a blocked airway through a solid overcast.

A trap-door in the floor of an overcast which would raise with the explosion and fall again with the heavy reaction that follows would enable the air-current to be reestablished and meet the objection to the air-bridge. Such a door could be built cheaply and can be made tight for ordinary emergencies.

The greater expense of the rock tunnel would result in a smaller number of them being driven, and hence less split air currents. As the economic distribution of air depends upon the use of numerous air-currents, there would be a greater economy secured with the same outlay by an abundant use of the cheaper air-bridges than would be obtained for a similar number of solid overcasts.

The mining laws of Pennsylvania require overcasts and air-bridges of masonry or in solid rock, in all mines generating fire-damp in sufficient quantities to be detectible by the ordinary safety-lamps.

An air-bridge passing over a slope is always built in the solid.

Mine-doors.—The doors are of two classes—those provided with an additional valve or gate, which may be opened or closed within a limited range; and those doors swinging on hinges, which may be opened for purposes of ingress or egress; they should open against the direction of a possible inlet current, in order to completely direct the current along the main way. Doors are the main dependence for the ventilation of mines, particularly those which are subdivided into districts for purposes of ventilation. They are placed in such position as will

temporarily and effectually check and deflect the current. Wherever their location, they must be built with great care, of matched timber, closely fitted in their frame, and are maintained only so long as the draught is to be kept up. Some are even provided with weather-strips on their edges. They are placed in all the side entries to the rooms, swinging outward towards the main passageway, and are generally in pairs when located in the haulage-way. In the latter case they are located far enough apart that the two will not be open at the same time, and thus interrupt the principal circulation.

Regulator-doors are employed (Fig. 168) at the mouth of entry to a ventilating district in the mine, the gate of which constitutes an adjustable sliding door capable of being secured against disturbances, and are employed to regulate the supply of air to be delivered to that district in accordance with the system described above. The placing of the slides is left to the mine-boss or the fire-boss. Those doors which are placed in haulage-ways either require an attendant or have their frames so inclined that the door swings to of itself. Automatic mechanical appliances for opening and closing the doors from a distance without stopping the mules or other haulage-motors are employed in many collieries, but do not stand in great favor. While there seems to be no means yet offered for replacing doors or dispensing with them entirely, there is no question as to their objectionable nature. They are leaky, and offer opportunity for negligent drivers, who by leaving them open divert the current from its proper course or stop it entirely. The most objectionable feature is their liability to destruction by explosion, and the consequent annihilation of the current at the most critical time.

The Effect of an Obstruction in the Airway.—Any irregularity in the sectional area of the airways or any fall of rock restricts the passage of the air. So the movement of the cages in the shaft will disturb the current, and the same occurs with the passage of trains. Their effect is to increase the mine resistance, which will be apparent at the fan in an increased speed, even

though the quantity of air being delivered may be less. If doors are left open in the mine, the reverse takes place.

When the number of ventilating circuits is large, the obstruction in one of the splits will have little effect on the fan or furnace as revealed by the water-gauge. The greater the number of splits, the less will be its influence. But a redistribution takes place throughout the mine, and all the other splits will receive an undue proportion of air.

The regulator-doors are then reset until the obstruction has been removed. The effect of placing a regulator in any district cannot be calculated without taking into consideration the whole circumstances of the mine. Any change in the position of a shutter affects the other airways.

Precisely the same formulæ will be employed for the equivalent orifice of the mine as will apply to mine regulators, the coefficient for the *vena contracta* being taken at 0.65.

FIG. 168.—A Regulator Slide-door.

Safety-doors.—In some collieries an excellent device has been introduced. This consists of a sheet-iron door hinged and suspended from the roof at appropriate places, to be released when the explosion occurs, drop, and close the opening, thus replacing those disturbed by the accident and maintaining the direction of the current at this critical time.

Brattices are used as temporary expedients for subdividing any room or airway in such a manner as to carry an inlet current through the wider compartment to the breast or face of the work, and return it in the smaller section to the return airway. Brattices may be made of planks nailed on props at suitable distances apart, with the interstices between the plank lathed, and the whole tarred or calked with oakum to constitute an air-tight partition, or the brattices may be made of canvas unrolled horizontally and suspended from the roof and frequently adjusted upon posts, according to the ventilating pressure in the airway or room in which they are placed. To make canvas impermeable to air, it is usually soaked with tar, though its stench has resulted in its substitution by an incombustible material, such as asbestos or a soluble silicate.

Aid to Ventilation.—Other methods of providing the return- or inlet-way consist in cutting a ditch in the floor of a gallery and boarding it over; in providing a "top sollar," the timbered roof of the gangway having a little extra space for the passage of the air-current above it instead of below it; in laying wooden air-boxes, metal-pipes, or large canvas hose at one side for temporary expedients.

In the rooms, entries, and shafts the distance between the working place and the last connection with the airway must not exceed 22 yards, and that from the end of the brattice must not exceed one fifth of that distance.

"Throughs" or "dog-holes" are driven at every 30 yards through the pillar coal to connect rooms, and to allow of air circulation through them in turn as the room advances.

For intensifying the permanent air-current led to a working place, or for the separate ventilation of workings in seams with slight disengagement of gas, compressed air may be employed or hand-worked fans may be used for separate ventilation, which is always brought up so near to the working place that the air be not too much diffused.

EXAMPLE.—A downcast $6' \times 11'$, an upcast $6' \times 12'$, 300 feet deep, supply air to a mine having a gangway 3000 feet long, 50 feet sectional area ($5' \times 10'$);

five splits receive each a portion of the 40,000 cubic feet moving. Required the several amounts delivered to the panels through the resistance of $6' \times 8'$, 700 feet long; $5' \times 7'$, 1000 feet; $5' \times 8'$, 1200 feet; $6' \times 6'$, 750 feet; and $4' \times 7'$, 800 feet.

By substitution it will be found that the pressure required to overcome the downcast resistance is 1.23 lbs. per square foot. In the mine the pressure will be the same throughout; hence for each split

$$q^2 = \frac{a^3 p}{flm},$$

which, solved for each division, gives volumes $q = 16{,}059\sqrt{p}$; $9030\sqrt{p}$; $9679\sqrt{p}$; $10{,}909\sqrt{p}$; and $7543\sqrt{p}$, respectively. The sum of these equals the total quantity, Q, from which we get p, equal to 0.87 lb. Assuming that the splits are made as near as possible to the downcast entry and reunion to upcast, which by the way is advised, there need be no further allowance for resistances other than those of sudden turns or contractions. The distribution of the air, q, for each district is 12,070, 6785, 7275, 8200, and 5670 cubic feet.

The upcast offers a resistance to the exhaust-air, the volume of which is greater than the 40,000 cubic feet, because of accretions from "blowers," moisture, etc. Disregarding these increments, the volume to be exhausted is carried up at a velocity of 555 feet per minute, whence p is 1.00 lb.

The total pressure to be imparted by increasing the downcast barometric or rarefying the upcast is therefore 3.01 lbs., requiring M of 57 feet, or a difference in temperature of about 100° at $B = 25$ inches. If the splits are made at stated distances along the main gangway, an allowance must be made for each of the several losses of friction in the various lengths thereof, remembering that each branch split reduces the volume passing through the remaining portion of the gallery, and correspondingly the friction therein. It would require 3.7 H.P. to do the work upon the air; a fan of 43 per cent efficiency would necessitate a 9 H.P. engine.

The calculation for the ventilation of a railroad tunnel is similar. Assume a tunnel 4961 feet long; sectional area, 336 square feet. Then $4961 \times 336 = 1{,}666{,}900$ cubic feet of air to be changed every ten minutes. Velocity of current, 496 feet.

$$p = \frac{flmv^2}{a} = 4.8 \text{ lbs. per square foot.}$$

A fan $20' \times 6'$ at forty revolutions easily meets this demand. The fan may be applied, with or without brattice, at either end of the tunnel, but this is a delicate matter. It would be better to place the fan at the mouth of, or a

furnace at the bottom of, one of the connecting shafts used during construction, and block the others off.

Required the pressure and power to get 10,000 cubic feet of air through airways aggregating 6300 feet in length, of which 5000 feet are along a heading $5' \times 8'$ in area, with 1000 feet of airway $6' \times 9'$ at the intake end and 300 feet of a $2' \times 5'$ air-course at the upcast end.

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CHAPTER XV.

THE ILLUMINATION OF MINES.

Illumination in Mines.—In an atmosphere free from explosive gas, the illumination may be had by any form of naked light. In all metal-mines candles are used, and, occasionally, the torch and kerosene-lamp. In bituminous mines known to be non-gaseous the latter is employed; but in all mines which are at all likely to develop gases the lamp-flame must be protected from direct contact with the white or fire-damp. The simplicity of the means of illuminating metal-mines tends to promote the comfort and safety of those engaged in them as the rays of light from the open candle or lamp are shed in all directions, thus revealing dangerous roofs, sides, or timbering much better than any form of safety-lamp can. But only in very shallow collieries should this be permitted, and then only after rigid inspection by the fire-boss.

Candles and Oil-lamps.—The candles which are used in metal-mines are usually of stearic acid, of which Proctor and Gamble's are the most uniform and will best withstand the temperature of the heated atmosphere. They are cheaper illuminators than lamps in rooms and stopes, but not in haulage-ways. The consumption averages three candles per man per shift.

The candle may be placed in a loop sewn in the cap or the pointed shank may be stuck into the timber, roadside, or floor, where it is shielded from the current of air. With a proper supply of pure air it burns without unpleasant smoke emission, and in places where there is a certain proportion of carbonic acid gas in the air, which would surely cause its extinction if left undis-

turbed, it may be kept alight by spreading out the wick so as to provide spaces between its cotton threads, and inclining the candle for the tallow to feed the flame faster.

The common tin lamp with the hinged lid on top and a hook and spout on either side—from the spout the wicking projects and is warmed—is a more brilliant illuminator, and is also used in coal-mines, giving a moderate light of about four candle-power, with, however, the objection that it smokes.

At many collieries the pit-bank and the main roads from the pit-bottom are lighted by electricity, or by gas, manufactured in the colliery yard, but where this is not so, larger and more powerful lights than those used at the face are necessary. The common arrangement is a can holding about 3 to 6 pints of oil, with one, and often two large wicks, having a tightly fitting lid and a chain carrying a pricker, for use on the burning wick when required.

Illuminants.—Vegetable or animal lamp-oils were at first used in safety-lamps, but now mineral oil is commonly used. But mineral oils are volatile *per se* without decomposition. Hence, even with heavy mineral oils having a flashing-point high enough to warrant their use in safety-lamps, great care is necessary in their storage and in the filling of lamps. Mineral oils are more perfectly fluid, more combustible, and give a better light than vegetable or animal oils, but unless there is a good supply of air to feed the flame they are apt, owing to the large amount of carbon they contain, to deposit much soot. Kerosene requires an admixture of a less volatile oil.

The two staple illuminants for safety-lamps are best refined rape- (colza-oil) and seal-oil. These give off no vapor under a comparatively high temperature. The highest value of these oils is only obtained from careful preparation, and cheap productions are to be studiously avoided. White lard, winter-strained oil, is also much used, the consumption being one half gallon per month for each lamp. In some mines a mixture of equal parts of seal-oil and petroleum seems best to meet the requirements of a good illumination with a minimum of smoke. In a mine

using 260 duplex-wick lamps the annual expense for oil, repairs, interest, etc., is $504. In selecting oil for illuminating purposes, its behavior is tested not only from the standpoint of its usefulness as an illuminant, but also that of its ability to burn without smoke. When the oil burns and the combustion is perfect, a blue non-luminous heating-flame is produced; but when the conditions are such that the flame is cooled during combustion or receives a deficiency of oxygen, the combustion is imperfect, and the portion of the carbon in the oil is rendered incandescent, thus emitting light. When the oil is very dense, the amount of incandescent carbon released becomes excessive, particularly in the presence of a small amount of oxygen, and soot is the result. The ideal oil, therefore, should furnish a maximum of light and a minimum of soot, with sufficient combustion to produce draught. A simple test and a decisive one may easily be made for the fitness of oil for use in the miner's lamp by burning it, under the ordinary conditions, in a common house lamp with a short chimney. The mixtures, which are often used, of mineral oil with animal and vegetable oil are always objectionable because of the almost unendurable odor, which itself is detrimental to good air. There is little saving in their employment, and they are worse than the unadulterated oil. The very volatile oils and spirits, like benzine, burn with a clear, uniform flame, show an easily perceptible cap in the presence of gas, and are usually very sensitive, being also free from danger in a well-constructed lamp, even in the hands of an unskilled miner.

The Davy Safety-lamp.—Much ingenuity has been expended in the endeavor to invent a safer means of illuminating workings than that offered by the naked flame. In 1815 Davy discovered that a sheet of iron-wire gauze was so good an absorbent of heat that the flame in contact with it could not readily pass through. Further experiments indicated that for mining purposes a mesh of 784 holes to the square inch was the safest, and was therefore adopted as the standard. A cylinder of this mesh, surrounding the light, surmounting an oil-lamp and

capped by a perforated top, is the form, which has been little changed since Davy's time (Fig. 169). After the lamp is filled

Fig. 169.—Davy Lamp   Fig. 170.—Clanny Lamp.

with oil and lighted, it is locked, to bar the miner against access to the flame, the wick of which is trimmed by a wire passing up through a close-fitting tube from the bottom. The combustion is supported by air penetrating the gauze at all sides.

Sir Humphry Davy thus describes his invention: "The principle of my lamp is that the flame, by being supplied with only a limited quantity of air, should produce such a quantity of azotic or carbonic acid gas as to prevent the explosion of fire-damp, and which, from the nature of its operations, should be rendered unable to communicate any explosion to the outer air."

Defects of the Davy Lamp.—This lamp has done and continues to do great service; but it has two defects. The first is the liability of the gauze to become red-hot, and allow the flame to pass through to the inflammable mixture outside. The second objection is its low illuminating power. The open spaces occupy only one fourth of the area of the gauze, through which the light escapes horizontally; still less light gets out at the top, to illumine the roof. Miners require light thrown in every direc-

tion, especially upward. The illumination of the roof is a more important matter than the lighting of the working face. It is also more difficult. The Davy lamp is inadequate for roof inspection. These defects have been partially remedied in the subsequent patterns by the use of glass, the only impermeable, strong, though brittle, transparent substance.

The Clanny Lamp (Fig. 170) is a modification of the Davy, in which a portion of the wire cloth is replaced by a short cylinder of glass. This improves the illuminating power, and, if enclosed in a tin can or shield, becomes quite safe in a gaseous mixture. The majority of the later forms of lamps use a glass cylinder, but an internal explosion, if it occurs, is rendered more violent than in the Davy, which offers less obstruction to the escaping gases.

Stephenson's Lamp is almost as popular in America as are the earlier forms, having a long cylinder of glass surrounded by a wire gauze, bonneted above by perforated copper. The air-feed is also through the gauze, passing underneath and into the chamber to the flame, thence out at the top, as usual. This plan keeps both cylinder and gauze cool, and its relative security rests essentially on the regularity of the draught, for if the inside air becomes overheated the light goes out; so it must be suspended properly. This is an English favorite.

The Marsaut Lamp is an improvement upon this form, and stands a fair amount of tilting safely. With care, its glass cylinder will last three years before breaking. The Marsaut lamp in many mines abroad is regarded as the most suitable one for the working miner, its construction being simple and strong. As an indicator of gas it is reliable, furnishing also a good light. In 370 in use, the average consumption of rape-seed oil was 2 gallons per year. This lamp was brought very prominently before the public by the Accidents in Mines Commission. A great difficulty is experienced in relighting it, and, from the winding path pursued by the feed-air, proper circulation does not take place until the lamp gets hot.

The Mueseler, a Belgian lamp, is like Dr. Clanny's, having in addition a conical chimney centrally above the flame. It is

highly recommended in Europe, but must be carefully handled. It does not burn well in "dampy" or slow currents. The bonneted Mueseler, an English improvement, is receiving the highest encomium for use in fiery mines and high velocity.

The Hepplewite-Gray Lamp admits air at the top, down four tubes, and through an annular chamber above the oil vessel. The only gauze employed is that covering the outlet and annular inner chamber. A serious difficulty with it is its liability to be extinguished when suddenly lowered. It undoubtedly gives more useful illumination than any other lamp, and as an indicator of gas undoubtedly ranks superior to all others—except, possibly, the Pieler or Wolf varieties. All other forms with the inlet above the glass will miss, say, 4 inches of gas lying immediately against the roof, except when they are tilted very much, and then there is great danger of their going out. Many lamps are now constructed to take air, if desirable, from the top, like the Gray, and thus also to detect thin layers of gas; but even then they will not do it so rapidly. It is possible to put some modern lamps into gas and take them out again without any indication being given—if the test is conducted hurriedly. This is quite impossible with the Gray, as the flame immediately "spires" up. Owing also to the large amount of useful light given by it and the way this is directed on the roof, in addition to its delicate indications of gas, this lamp is preferred to all others for use by deputies, firemen, timberers, and fire-bosses.

The Dick Patent Port-hole Lamp compels all the air entering the lamp to go immediately to the flame, thus losing no air, and is capable of burning in a stagnant atmosphere. The air entering the lamp above the case passes through the gauze, thence descends to the flame, while the products of combustion arise inside the lamp, to be emitted through circular holes at the top of the bonnet. The bonnet is made of a seamless steel tube, and is light and strong.

The Clifford Lamp is a new one and has many excellent points. The light given is good. A plentiful supply of air enters the lamp. It illuminates the roof as well as the sides of

the workings. It is not sensitive to tilting or violent movement. The bonnet arrangements prevent the possibility of a current of any velocity from impinging directly upon the gauze, and thereby render the lamp safe in explosive currents.

The Beard-Mackie Lamp has a small brass disc supporting a Ω-bent rod, across whose standards are strung platinum wire. The height of the flame is made visible by rendering the wire incandescent. The percentage of gas present is known from the height of wire rendered luminous. As the wires can be altered in position by the mine-boss, the warning line can be regulated according to the risk to be run. The lamp is enclosed in a wire gauze or in a bonneted glass cylinder.

The Woolf Benzine Safety-lamp is an emphatic departure from the varieties above described, in that, first, it burns benzine or naphtha; second, it contains a patent self-igniter capable of relighting the lamp fifty times without opening; and, third, it contains a locking device which it is impossible to open except by the use of an exceedingly powerful magnet. This lamp, because of the sensitiveness of its illumination, is a delicate detector of gas, and has met with very ready acceptance throughout coal-mining districts, there being possibly 80,000 in use in Germany. It furnishes a good indication of the presence of gas by the height of its flame. A percentage of light carburetted hydrogen, varying from 1 to 5 per cent in the total mixture, will increase the height of the flame from 2.6 to 5.8 cm. If the flame be partially turned down, a still greater degree of sensitiveness will be manifested.

The Requisites of a Safe Lamp.—The lamp must be self-contained, strong, portable, and not heavy, require little attention from the miner during twelve hours of sustained light, and capable of being placed in any position, besides giving perfect insulation from the fiery gas. The use of the safety-lamp is to secure protection to the miners from explosive mixtures, whether by allowing of safe examinations by the fireman before the shift begins, or by preventing explosion from sudden eruptions or ordinary discharges of gas during the shift. It should afford

good illumination of the roof during repairs to the latter, or during ordinary operations of mining, without tempting the miner to uncover the flame. The wire-mesh casing reduces the photometric value of the light to a low degree.

The illumination from any of these lamps is very feeble. It is less in any direction than the horizontal. Of all the lamps the Gray sends the best light upward. The candle-power, horizontally, of the Roberts is highest—about 18, and of the Clanny the lowest—nearly 6. On this account a lamp must be able to be held tilted without extinguishment, and be unaffected by violent oscillations. The conditions dictated by safety circumscribe the lines of attempted improvement in the degree of illumination. For, with a large mesh, the lamp is incapable of preventing high-speed gaseous currents from entering the construction chamber and blowing the flame against the opposite gauze, or forcing gas into contact with the flame. The safe lamp must have a mesh impermeable to the high-speed currents. Glass improves the illumination but introduces other dangers—risk of breaking, if unannealed, by cold-air current or dripping water, and the difficulty of securing a good joint to the gauze cylinder.

Of the forty-one explosions which occurred in a certain district during 1896, in four cases the immediate cause of ignition was referred to a naked light or to a deterioration of the safety-lamp; in twenty-five, to the passing of the safety-lamp flame against the gauze through some careless movement, too high a speed, or "falls in." The remaining twelve were from shot, fire, or other undetermined causes.

Designers of lamps have difficulties to overcome other than these. They include the rough usage lamps receive in the mine, the violent shaking and tilting they are subjected to, frequently causing discharges from the oil-chambers; the dust finely suspended in the air, or settling on them at the faces, making them more explosive; oblique explosive currents at the face caused by falling stone or coal, close to the lamps; and the grimy hands of

the collier, which clog the gauzes and inlet feed-holes with dirt and grease.

An absolutely safe lamp is therefore difficult to obtain, and notwithstanding the various modifications, there is as yet none. The lamp which cannot ignite in an explosive mixture outside of it is yet to be invented.

The defects of the Marsaut lamp are at the metallic connection of the bonnet with the other parts. They heat up from the impingement of the flame and gases. Gas moving at 50 feet per second can penetrate the two-gauze Marsaut, but a three-gauze pattern will resist the current. The only source of danger in the Mueseler lamp, not common to the other types, is its strong tendency to smoke; otherwise it is a most efficient lamp, showing a bright and steady flame in the strongest current. The Gray lamp presents the risk of the gas burning at the cylindrical strip of gauze under the glass, which in low-speed currents heats the lower edge of the glass strongly, and in high-speed currents allows the heated gases to pass across against the glass on the other side. The top of the lamp can be easily tampered with, and the gauze at the outlet is liable to be obstructed by soot if the flame should smoke.

It is worthy of note that the brass lamps are less bright than the iron lamps of the same pattern. A round wick is not so luminous as a broad, flat wick for a given oil consumption. Seal-oil is photometrically better than rape-seed-oil illuminant. The insufficient light of a safety-lamp, combined with the difficult and trying conditions of the bonneted forms, is proving injurious to the eyesight of miners, which serious evil is growing. Photophobia is rare where candles are used, or where the lamp is hung behind the miner.

Bonneted Lamps.—In order to be safe in the highest velocity of air-currents within a given mine, the flame must be enclosed not only in a wire gauze, but also in a more or less impermeable hood or bonnet, while the inlet area for the feed-air must be reduced to the smallest allowable dimensions. Many lamps now exist which appear to resist, in a highly explosive atmos-

phere, current velocities up to 3000 feet per minute for a period of several minutes; and the four lamps which were brought to the attention of the Mines Accident Commission, which received special attention for their security, illuminating power, and simplicity of construction, were the H.-Gray, Marsaut, bonneted Mueseler, and Thomas's modification of the bonneted Clanny.

The bonnet screens the gauze cylinder from the effects of draughts that blow the flame through the meshes and set up a fiery heat by the excessive air and gas that enter above the flame of the wick. It limits the supply of air to that required for the oil-flame only. Such bonneted lamps, whose flames are protected from the direct effects of the strong ventilating current, may be used with safety for illumination in mines producing fire-damp. Even in dry, dusty mines also developing fire-damp some of these lamps are safe, though not all, for many well-authenticated cases of failure are recorded where the dust has proven fine enough to pass through the gauze meshes, to be reduced to the state of incandescence in the inner chamber. Locked safety-lamps are insisted upon in mines when cutting through clay-veins in solid workings. The Hepplewite-Gray and the bonneted Mueseler have the best resistance to explosive currents of high velocity, and the South Side Committee report the following relative speeds at which the respective lamps and the air-current can safely pass: Davy, 360 feet per minute; Clanny, 600 feet; Stephenson, 780; Mueseler, naked, 1200; Mueseler, bonneted, 2400; Marsaut, in a can, 2440; and the Davy, in a shield, 2400. The North of England Institute of Mining Engineers gives the safe velocities at 720,540, and the others higher. The British Royal Commissioners of Accidents approved the Gray, Marsaut, and the bonneted varieties as safe at high speeds. The common Davy or Geordie lamps are unreliable.

Locking Lamps.—All safety-lamps have locks to them. The oldest form of lock is still used on most lamps, notwithstanding its acknowledged insecurity. It consists of a screw-bolt with a square head which is turned by a key until it has entered a hole bored in the oil-chamber, which is supposed to prevent

it from being unscrewed until the screw-bolt is withdrawn. But this may be done by a couple of nails filed to fit the bolt, by pieces of wire bent for the purpose, or even by the point of a pick. It affords a little more security than the original device of the screw-lock.

The importance of locking the lamp so that its flame cannot be exposed to the gas is manifest, as there are many temptations to the miner to open it. Most of them can be, and are, opened by easily extemporized mechanical means, while others are rather more difficult to open.

All manner of permutation-locks and magnetized plates are offered on the market, besides the lead-plug seal with which the lamp is riveted after each filling. On account of its simplicity and ease of treatment, as well as from the measure of security it affords, the lead-pin is coming more and more into use at collieries. The pin is moulded with a head at one end, and fits openings in the two parts of the lamp to be held together. When in place it is firmly riveted and punched with some device at both ends. This forms the means of detection, if the lamp should be wrongfully opened.

Magnetic locks are employed to advantage. Lamps so fitted cannot be opened except with the aid of a powerful magnet. They resist all the efforts of the miner, short of wreckage of the lamps. The Woolf lamp and the Craig & Bidder's lamps are so fastened.

Extinguishing-locks. These are devised to remove all temptation from the collier to unlock his lamp by so contriving it that the process of unlocking also extinguishes the flame.

Lighting and Relighting Locked Lamps. — A great many lamps become extinguished at the working faces from a variety of causes, and this has led to designs of lamps which may be quickly relighted without their being unlocked. The designs may be divided into two kinds, in one of which lamps may be relighted anywhere by the user; the other, only by an authorized person at properly appointed stations by the application of electricity.

Woolf's lamp belongs to the first class, the relighting arrangement being adapted for the volatile illuminant, benzoline, used in it. The lamp contains a reel of paper on which fulminating spots are placed at intervals of about a quarter of an inch, each of which can, in turn, be brought opposite the wick, and at the same time be struck by a small spring-hammer, operated by a button at the bottom of the lamp; the composition explodes and ignites the benzoline vapor surrounding the wick.

Use of Lamps.—Safety-lamps provided with the best form of lock to prevent their being tampered with by ignorant or reckless workmen, and thoroughly tested before given out, will, if carefully used, afford considerable protection in the mine. No lamp is safe unless kept in thorough repair, and any infraction of rules regarding careful use should be severely punished.

Examining and Testing Lamps.—The lamp-room is on the surface, where it is safer than underground. The safety-lamps are complicated in their construction, and the work of examining and repairing them in large collieries is considerable. As it is difficult for the eye to detect defects in adjustment, there should be provided some simple means for reliably testing the joints between the glass cylinder and the gauze. It is well to test each lamp, before it is taken into the pit, by placing it, lighted, in an explosive mixture.

Cleaning Lamps.—At small collieries having a few safety-lamps in use the cleaning is usually done by hand, without the aid of mechanical contrivances to unscrew or remove the internal fittings; an ordinary hand-lamp brush is used to rub the dirt from the gauzes. Even when the workmen clean their own lamps there is at least one lamp-keeper at each colliery to replenish the oil vessels, renew the wicks, and replace worn washers or broken glasses, except in the very rare instances in which the men themselves supply all safety-lamps and afterwards maintain them in a perfect state of repair. Where oil is burned the gauze should be steeped in a hot alkaline solution, to free it of soot, etc. Lamps burning benzine are not clogged with carbonaceous deposit.

To avoid waste, manufacturers furnish automatic fillers holding the exact quantity sufficient for one lamp.

Electric Lights in Mines.—The advantages of electricity for lighting about a mine are its decreased cost, better illumination, absolute reliability, and greater freedom from accident. The lights outside are arc lamps; those inside are incandescent. Arc lamps are either of the open type, or enclosed. The latter give less light under similar conditions, but their distribution is more even. The former require from 8 to 10 amperes, and the latter 5 amperes of current. Though the efficiency of the latter is less, their consumption of carbon and the expense of their renewal are also less.

The lamps may be connected continuously in one series, in which case the dynamo is series-wound. If the lamps are in series, each one is provided with an automatic cut-out allowing the others to receive the current in case of its failure. When connected in several short circuits from the one main, "in parallel," they will require a compound-wound machine. A resistance is then put into series with each circuit in order to keep its current at the proper pressure. The latter connection is more common for mining work.

Each circuit is provided with a fuse which bears the same relation to the installation that the safety-valve does to the boiler. Fuses are made of alloys of tin or lead for the 50-ampere circuits, and of copper for heavier lines.

The light wires are protected from damp by being inserted in cast-iron piping or lead tubing. A double-pole switch mounted conveniently on a metal base, easily reached, opens and closes the circuit for the lamps. Having two contacts, it divides the current and the spark, and thus reduces the risk.

Lamps of 16 candle-power on a circuit of 100 volts require 0.6 ampere. On the 220-volt circuit, 0.27 ampere. These correspond to 3.75 watts per candle-power.

12 lamps of	16 C.P.	require	1	H.P.
6 " "	32 C.P.	"	1	H.P.
2 arc lamps of	600 C.P.	"	1	H.P.
2 " " "	1000 C.P.	"	1.7	H.P.

It is usual to allow 30 candle-power for every 100 sq. ft. of floor area, when they are placed to advantage. A radius of 70 feet is allowed for each 5-ampere enclosed lamp or a 10-ampere open lamp.

For parallel distribution of current to lamps, let w = watts per candle-power; W = watts per lamp; E = voltage at the lamp terminals. Then $I=\frac{W}{E}$ = the current in amperes per lamp. All wiring is based on an allowable drop, of which 5 per cent may be considered a good average for the voltage loss in this character of wiring. The area of a conductor to furnish the current for a given number of lamps, N, is

$$A=\frac{10.81\times 2D\times I}{\text{drop in volts}}=\frac{10.81\times 2D\times N}{R\times \text{per cent loss}}.$$

R is the heat resistance of one lamp.

Example.—It is desired to illuminate the grounds and buildings about the tipple by 100 16-candle-power lamps on a line aggregating 600 feet from the generator, 30 lamps of 32 candle-power, each at a distance of 750 feet, and 6 arc lamps in pairs at a distance of 600 feet from the power. Required the electric horse-power and the size of conductor, the voltage of the incandescent lamps to be 200 and of the arc lamps, 100.

As each lamp requires 0.3 ampere for 16 candle-power, 0.6 ampere for 32 candle-power, and 5 amperes for the arc lamp, the currents will be 30, 18, and 30 amperes, respectively. For the small lamps 600 feet of line must carry 30 amperes. This will require the wire of No. 4 B. & S. gauge having a resistance of 1.5 ohms per mile. The loss on this line will therefore be 45 volts per mile and 5 volts for the line.

The larger incandescent lamps will be on a cable of No. 6 B. & S. gauge having a resistance of 2.5 ohms per mile or loss of 43.5 volts per mile; and 6.2 volts for 750 feet.

The dynamo should deliver a current at its terminals of 2.06 volts pressure and 78 amperes. This equals 16,068 watts, which at 85 per cent efficiency correspond to 26-brake horse-power.

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CHAPTER XVI.

ACCIDENTS IN MINES.

The Inherent Dangers in the occupation of mining are great. The uncertainties in the overlying ground, the ever-lurking foe in the form of gas, the as yet imperfect methods of illumination, demand the greatest vigilance as the price of comparative safety. Add to these the ever-present factor of human carelessness, and the equation becomes one, containing two independent variables, the solution of which is hard to find. To what extent the inherent dangers could be overcome, were the factor of carelessness eliminated, it is difficult to say. The German Government, out of a total of 7933 lives lost in coal-mining, attributes 5179, or 65 per cent, to the purely inherent dangers of the occupation; 2397, or 30 per cent, to the carelessness of the victims; 248 to the fault of fellow workmen, and 69, or 0.9 per cent, to defects in working. Reports of mine inspectors attribute 60 per cent of the deaths in mines of the United States to carelessness; the 40 per cent to unavoidable or unforeseen causes.

Comparative Hazard in Nations.—Under this head it is impossible to obtain absolutely accurate conclusions, owing to the great difference in methods in vogue in different countries. In Europe the record of accidents, injuries, and fatalities has been much more elaborate and precise than in America, while the classification of accidents varies in the different countries. In France an accident is not entered on the records unless the person is prevented from working for a period of twenty days; in Austria it is the same; in Belgium eight days is the limit; in Germany the law is still more strict, as it is necessary to give notice of inability to work for any period exceeding three days.

(Every detail accompanies the inspectors' reports in Europe, such as the age of victim, nationality, character of work, length of time in service, exact location of accident in mine, nature of injury, character and extent of medical attendance, length of time incapacitated, number of days absent from work due to injury. Even family relations are reported, as number dependent upon the man, etc. In America inspectors' reports are as a rule more general or indefinite, due to the different conditions and consequent differing laws existing in the several States. As yet we have no Federal requirements save those locally applied in territories and reservations.)

There are two methods of comparison of hazards: 1st, classification by rate of accidents per number of tons of coal produced annually; 2d, by rate per accident per number of employees working underground, or by total of surface and underground laborers. If the estimate be made on the first basis, allowance must be made for the greatest efficiency of the American laborer, due largely to the shallowness of our coal-seams and the extensive use of machine cutters. In the table below can be seen the productiveness of the American compared with the English and the Belgian miner.

DEATH-RATES AND TONNAGE PER EMPLOYEE.

	Tonnage per Employee.	Tons of Coal per Death.	Rate per 1000 Employees.
United States, 1893 to 1902, both inclusive, bituminous	660	221,094	2.89
" " " " " " " anthracite..	381	126,795	3.81
Ohio, " " " " "	514	292,596	1.76
Austria, 1894 " " " "	326	199,526	1.718
Belgium, 1896 " " " "	178	123,358	1.01
France, " " " " "	207	182,525	1.23
Great Britain, " " 1903 " "	314	238,042	1.31
Prussia, " " 1902 " "	338	150,000	2.16

The second method of classification is generally accepted as the more reliable and is more readily understood. It also affords a comparable standard of comparison with other trades. The following table requires little explanation. It represents conditions in the three prominent coal-producing

States of America. The utmost care has been taken to insure the accuracy of all data used, and all statistics have been officially verified, as a thorough and comprehensive knowledge of the dangers and fatal consequences in this occupation is a matter of absolute necessity to those engaged in it.

1902.	Pennsylvania.		Illinois.	Ohio.
	Anthracite.	Bituminous.		
Tons of coal	36,911,554	98,946,204	30,021,300	23,929,267
Number of employees inside	98,377	110,015	41,518	27,613
Number of employees outside	49,762	25,371	44,487	9808
Fatalities, total	300	*456	99	81
Serious injuries	641	861	406	298
Deaths per 1000 employees	2.25	3.36	2.1	2.1
Injuries per 1000 employees	4.32	6.36	8.8	7.9
Number of days worked	220	116	210.4	176.4
Tons of coal per life lost	123,039	217,000	303,245	293,818
Annual output per employee	376	898	724	867

* Includes an explosion involving 112 lives.

The Death-rate in Mines.—During the earlier period of mining the loss of life annually was at a rate of 1 death for every 200 persons employed. With time this rate is appreciably reduced. In Great Britain, during 1903, it was 1 for every 688 employees. The average in the United States is now about 2.3 lives for each 1000 men employed, or 1 for every 435. During the year 1902 the total mineral output of the United States required nearly 1,000,000 employees, of whom at least 2500 lost their lives. In Great Britain, France, Belgium, and Prussia, during the same year, coal-mining, in producing 478,459,088 tons of coal, killed 1937 below and 306 at the surface. Metal-mines are more hazardous than coal-mines, but in either the risk may be said to be less than in other of the dangerous occupations. The annual rate of death among railway brakemen during the past decade was 12.5 deaths per thousand; that among trainmen, 9.8 per 1000. The highest rate in any coal district to date is 8.5 per 1000; the anthracite coal-miners is 5.6, and among the anthracite laborers 4.6 per 1000. In the United States and Canada, for the decade ending in 1902, there were 21,150 deaths, a rate of 2.87

for each 1000 employees. In the entire railroad service during the same time the average rate of death was 2.64 for each 1000 employees. Among the bituminous coal-miners the rate averaged 2.2. The accident liability inside the mine is, of course, greater than among the laborers at the surface.

Statutory Provisions Properly Governing Underground Operations, to provide safety to miners, require: A safe ingress and exit, with ample means of communication between the bottom and the top; the examination of the working places by a foreman prior to the entry of the men; copious volumes of air in well-directed currents to dilute the accumulations of gas; secure timbering and effective illumination of the working places; the posting of copies of rules, advice, and precautions, printed in several languages; official inspection; an ample supply of timbers; large pillars to prevent caves of roof; supervision in the use of explosives in each working place; boundary pillars between gaseous sections of a mine and drowned or abandoned mines. Faithfully executed, these provisions are ample to protect the men from the extreme underground dangers.

It cannot be denied that the American death-rate is higher than in European countries, nor does it show the strong downward tendency that is apparent in their statistics. A fair example may be the record of Great Britain for the semi-decade since 1857. The annual death-list showed nearly 1000 lives sacrificed in 1852, and it is a striking fact that not many more were lost in 1902, but the number of employees have increased during that time from about 200,000 to 850,000. The improvement in the conditions of English mines is also manifest by the following summary, showing the tons of coal produced for each life lost:

1852	59,800 tons
1862	82,500 "
1872	112,000 "
1882	145,000 "
1892	200,000 "
1902	227,280 "

It is true the statistics of the States include only the coal-mines, though it is improbable that the inclusion of the metal-mines would bring the rate even to that of Great Britain, particularly as there are many small coal-mines not subjected to official inspection whose records are never kept. America is behind the other nations in the efficiency of its safeguards, or, at least, in the results.

AVERAGE NUMBER OF ACCIDENTS IN MINES IN THE UNITED KINGDOM IN SEMI-DECADES.

Periods.	Number of Employees.		Total Deaths.		Annual Death-rate per 1000 Employees.						
	Below.	Above.	Below.	Above.	Explosions.	Falls of Ground.	In Shafts.	Miscellaneous.	Total, All Causes, Below.	Total Above Ground.	Total Above and Below.
1851-55	182,427	47,047	937	47	1.280	2.016	1.206	0.556	5.149	1.012	4.301
1856-60	208,763	53,832	964	53	1.234	1.846	0.890	0.648	4.628	0.994	3.883
1861-65	237,779	61,314	898	68	0.618	1.714	0.668	0.790	3.791	1.105	3.240
1866-70	269,813	69,574	1071	87	1.158	1.578	0.528	0.730	3.995	1.256	3.433
1872-75	399,397	111,584	1066	99	0.516	1.210	0.437	0.572	2.736	0.899	2.342
1876-80	424,586	117,876	1147	100	0.811	1.132	0.317	0.449	2.709	0.847	2.306
1881-85	443,502	116,688	1025	99	0.408	1.108	0.263	0.532	2.312	0.848	2.007
1886-90	477,633	126,654	975	117	0.312	1.015	0.196	0.517	2.042	0.913	1.806
1891-95	571,463	150,804	979	123	0.281	0.806	0.194	0.934	1.706	0.822	1.531
1896-03	618,507	150,404	896	125	0.17	0.764	0.144	0.457	1.451	0.801	1.381

Each American State reveals continuous improvement in the underground hazard, but for the entire United States the death-rate has not been reduced. The reason for this is that new coal-fields and new mines are opened from time to time. The new difficulties and the increase of unskilled labor naturally increase the rate for the district. This is apparent in the new Western coal regions, where the rate is greater than in the well-explored Eastern districts. Indeed it is inordinately high.

Discipline in Mines.—Unquestionably a more rigid discipline in the operations of mines, a general education of the men, and a wiser discrimination in the selection of workers is required, and will contribute greatly to the safety of mining as an occupation. It is true that many causes of accidents are unforeseen and un-

avoidable, but it is also true that the largest percentage of the causes of accidents are those of a preventable nature, largely in the hands of the miners themselves. If a code of rules were officially issued dealing with the severe forms of mine disaster, the first step would be taken toward greater security. Again, the present laws err in not placing direct responsibility for defective conditions upon some one. If this were remedied, better conditions would prevail. Not only should the employers be

ACCIDENT STATISTICS OF NORTH AMERICA, 1893–1902.

State.	Deaths.	Rate per 1000.	Tons Produced per Death.	Tonnage per Employee.
Alabama	353	3.00	194,870	500
Colorado	413	5.59	110,000	592
Illinois	813	2.21	276,347	607
Indiana	145	2.57	365,518	938
Indian Territory	239	6.22	67,700	421
Iowa	212	2.34	221,849	518
Kansas	177	1.82	276,271	378
Kentucky	118	1.35	360,000	485
Maryland	71	1.56	61,366	957
Missouri	135	1.76	218,000	383
New Mexico	105	7.61	83,362	634
Ohio	540	1.76	292,596	514
Pennsylvania, anthracite	4344	3.01	126,795	381
" bituminous	2218	2.30	287,691	660
Tennessee	422	6.01	71,118	437
Utah	237	24.75	32,236	764
Washington	257	7.43	69,232	514
West Virginia	911	3.73	184,413	687
British Columbia	321	9.83	68,138	669
Nova Scotia	119	1.96	219,266	433
Total North America	12,150	2.89	187,384	541

fined for infringing or ignoring the laws, but the employees should be penalized for hazarding life and property of others.

The Causes of Accident.—The following prominent causes of underground accidents are classified according to frequency and number of lives involved. Falls of roof or sides of rock or coal; accidents in the haulage-ways; injuries in shafts; premature explosions of powder and explosions of dust and gas. The first three classes of accidents rarely cause more than one death

or a single injury at a time. Explosions of gas or eruptions of water, fortunately rare, have involved hundreds.

Falls of Ground.—This form of accident arises from the fall of roof or of coal in working places and in roadways. About 70 per cent of all accidents arise in the rooms, and about 30 per cent of the deaths occur in roadways. Each accident that occurs involves but a single fatality or injury. Yet it takes the lead as the most prolific cause of underground fatalities. The grave danger lies in the fact that the accident is local and of continual

FATAL ACCIDENTS IN THE PRINCIPAL COAL-FIELDS OF NORTH AMERICA.

States: 1893 to 1902.	Aggregate Number of Persons.	Total Number of Lives Lost.	Mortality per 1000.	Tons of Coal per Death.
Pennsylvania, bituminous; Ohio, Maryland	1,316,608	2,829	2.15	296,658
Indiana, Illinois, West Kentucky	456,341	1,017	2.26	293,874
Missouri, Iowa, Kansas, Indian Territory	303,147	763	2.52	170,000
Tennessee, West Virignia, East Kentucky, Alabama	457,475	1,745	3.79	165,636
Colorado, New Mexico, Utah	97,296	755	7.77	79,536
Washington, British Columbia	67,263	578	8.59	68,245
Novia Scotia, 1887–96	60,716	119	1.96	219,266
Total North America, bituminous	2,758,756	7,806	2.73	228,985
Pennsylvania, anthracite	1,443,110	4,344	3.01	126,795
Total North America	4,201,866	12,150	2.89	187,384

occurrence at a place where the men are otherwise engaged at work, and ignore or fail to notice the warning which usually precedes disaster. The weak spots in the roof, the various fossil tree-trunks, horses, balls of ironstone, rocks, naturally creviced, or shaken by the vibration of a neighboring blast, are liberated with little warning. To these, with the coal scaling from the sides or roof, the unnoticed yielding of pillars which are too thin to support the rock, and the underholed coal, the accidents are due.

Of the total fatalities in the anthracite region of the United States during the past twenty years, 3521 were caused by fall

of rock and roof in the working, or over 50 per cent of the total deaths during that time. In bituminous regions the proportion is equally large during the same years. The mine inspector for Illinois reports for the twenty years prior to 1903, a death-list of

CLASSIFICATION OF DEATHS FOR 1902. PRINCIPAL CAUSES.

Deaths from	Pennsylvania.				Illinois.		Ohio.		Kentucky.
	Bituminous.		Anthracite.						
	Deaths.	Injuries.	Deaths.	Injuries.	Deaths.	Injuries.	Deaths.	Injuries.	Deaths.
Falls.	223	437	116	256	59	203	54	162	5
Cars.	47	239	42	133	15	125	10	83	5
Gas.	126	20	20	59	4	15	3	3	..
Powder.	8	41	..	67	13	22	5	13	6
Surface.	14	36	55	103					
Totals.	456	861	300	641	99	406	81	298	19

THE STATISTICS OF ACCIDENTS IN THE PROMINENT COAL-PRODUCING COUNTRIES OF EUROPE, AND TWO AMERICAN STATES.

Countries.	Periods.	Persons Employed.	Tons of Coal.	Death-rate of Persons.				Total Above and Below.	Total Number of Deaths.
				Falls.	Cars.	Explosions.	Powder.		
Austria.	1894–03	125,303	367,088,336	0.282	.586	.310	.044	1.718	1,937
Belgium.	1896–03	136,880	170,979,816	....	....	...	....	1.01	1,386
France.	1896–03	168,609	243,671,927	....	....	..	....	1.23	1,335
Great Britain	1896–04	774,911	1,939,879,428	0.713	.125	.114	.042	1.31	8,190
Prussia.	1896–03	443,995	1,049,557,783	0.98	.68	.13	.11	2.16	6,948
Penn., anth	1882–91	1,059,526	351,058,927	1.37	.67	.29	.27	3.22	3,415
	1892–01	1,423,607	455,941,943	1.43	.59	.27	.30	3.11	4,427
Penn., bit.	1893–02	963,767	638,213,809	1.42	.34	.24	.06	2.26	2,184
Illinois.	1893–02	368,607	223,645,612	1.28	.21	.38	.08	2.25	813

1392, of which nearly 50 per cent is due to falls. Doubtless all over the United States the number of lives lost from this cause bears a similar proportion to the total. The death-rate from falls of roof and coal alone in the States is far more than is shown by the European statistics for the five years ending with the year 1900: France, 0.58; Great Britain, 0.78; Prussia, 1.22; Illinois,

1.34; Pennsylvania (bituminous), 1.84; and Pennsylvania (anthracite), 2.11. In Great Britain 22,190 lives, or 46.5 per cent, were lost by falls since 1853.

The death-rate is large and is not decreasing proportionately with other causes, for the conditions in the working places depend entirely upon the miner himself, and here improvement is not manifested. It is true that the pressure of the roof, and the dislodgement of the fragments therefrom, cannot be indefinitely resisted or avoided, but its movement can be detected and checked by systematic timbering. The best means of detection is the use, near the face, of wooden props, not iron, whose bending will give warning of disaster by buckling. A better illumination will reveal the condition of the roof, continual testing would disclose its loose fragments, and a systematic timbering would materially diminish the risk of its fall. The use of coal-cutting machines would remove the dangers of underholing coal; and so would the enforcement of a specific order for the dismissal of any employee failing to utilize the props. An accident is the result of deliberate neglect, and the delinquent is more frequently the old hand than the newcomer.

Rigid supervision and the appointment of a suitable timber-boss to visit all the working places and regulate the timbering, with power to punish carelessness, would be a most effective remedial measure. In certain districts in France the rigid enforcement of regulations requiring systematic timbering immediately upon advancing beyond the distance fixed by the regulation has reduced the loss of life to about $\frac{1}{8}$ man per 1000 from the previous rate of about 1 in 1000. In 1870–1879 it was 0.90 per 1000; to 1886, 0.24 per 1000; to 1890, 0.15 per 1000; to 1892, 0.13 per 1000. In Great Britain the rate of decrease during the same period from 1.20 to 0.764, though a decided improvement, was not so marked.

Accidents Due to Cars, Shafts, etc.—Second on the list of accidents in all classes of mines are those occurring in the travelling-ways. The men fall from, or are run over, by cars. Such accidents are more numerous in coal-mines than in metal-mines,

because the speed of the cars, or trains, and the number of men employed are greater. Many casualties arise from jumping on or off the trains while in motion. These are manifestly invited by the victims themselves; other than these, the accidents along the travelling-ways are those arising from runaway cars, or the lack of clearance in haulage-ways. Numerous safety-niches along the line will afford the men some protection. Those occurring in the shaft are largely averted by a security of the hoisting appliances, an efficient system of signals, and the use of overwinding and safety attachments. An average of 5 deaths from overwinding occurred annually in Great Britain. Cases in which men are caught between the cage and the shaft-timbers or are struck by material falling from above are purely accidental and may be classed as inherent to the occupation. Those occurring from the fainting of overheated men while being hoisted from the shaft are not numerous. These can be prevented by gates at the sides of the cages and hoods overhead. The European shafts are more fruitful of accident than the shallow American shafts.

Accidents from Use of Explosives.—In metal-mines the percentage of injuries due to premature blasts is larger than in coal-mines. They are more frequently the result of carelessness arising from the use of iron bars while loading a charge, from tampering with the metallic caps, handling powder near an exposed flame, thawing out frozen dynamite, drilling into unexploded cartridges, attempting to ignite too many blasts at one time, and returning to the work too soon. These accidents are manifestly local in their nature and occur either at the working-face, or at the powder-house where the explosive is stored. The victims themselves are the active cause. The comparatively high death-rate in the anthracite coal-mines is attributed by some inspectors to the great consumption of high explosives.

The use of electric cartridges and the prohibition of loose powder for firing in dusty collieries would materially diminish the number of casualties. Greater care in the selection of the explosive and its use can readily be exercised by both the em-

ployers and the employees. In some mines blasting is assigned to a special class of men.

Gas Explosions.—No coal-mine is free from gas, and no colliery can be operated without illuminating and blasting agents. Every mine, therefore, is subject to ignition of a greater or less volume of gas. The result may be a fire or an explosion, according as the amount of air present be excessive or at the critical percentage, and its effect will be proportional to the volume of gas or the quantity of coal-dust present. All dry mines should be ranked as fiery whether gaseous to a dangerous degree or not.

A steady flow of escaping gas into the regular working places rarely results in dangerous conditions, for the volume of the air-current there dilutes it to an innocuous degree. But when some internal reservoir of gas bursts into the entries or abandoned rooms, where the circulation is sluggish, or the air comparatively small in volume, the mixture of gas and air may easily become critical. This danger is imminent in the driving of galleries far in advance of faces in the rooms or through a seam beyond a fault where heavy volumes of gas may be liberated, particularly if the ground slip or be soft. A blast or a flame in contact with the mixture may ignite it and cause disaster. One gas explosion involving 62 lives occurred within twenty-four hours after the visit to a mine regarded by the chief inspector as "the safest and best-conducted mine in the State," but caused by striking a clay-seam. The ignition came from either a defective lamp or an open lamp. This is usually the direct cause. Men will disregard the prohibitory notice of the fire-boss barring them from a given working place, and enter. For one cause or another they will uncover the flame of their lamp. The reports for Great Britain show that out of the total of gas or dust explosions about 70 per cent owe their injuries to the use of naked lights or imperfect lamps; 18 per cent to shot-firing, and 14 per cent to the accidental, or spontaneous, ignition of the mineral. This form of accidents during the last twenty years has caused 1451 deaths, or 0.35 out of every 1000 coal-mines in Pennsylvania. It was 9 per cent of the total fatalities. Explosions killed over

10,000 men during the past fifty years, but the rate is decreasing very much, as shown in Great Britain for the successive semi-decades between 1852 and 1903.

Remedial Measures.—The measures which will safeguard the men from the consequence of sudden outbursts of gas include, besides the dilution of the gas in wide galleries, the use of numerous bore-holes to indicate an approach to a gaseous mass. This practice is sharply criticised by many because of the difficulty of boring in shattered, and consequently infested, portions of the seam. Some mines dispose of the gas through bore-holes, by which it is discharged to the surface and burned. An increased volume of air, its better distribution to working-faces, and the discontinuance of heavy blasts of explosives while the men remain in the mine are diminishing the number of injuries and fatalities from explosions. There remains much to be done by the universal employment of some bonneted lamp which is safe and at the same time furnishes good light.

A number of deaths charged to explosions occur in the exits of the mine where the fleeing victims have been overwhelmed by after-damp. Rescuing parties find them generally in the intake. Here it is that the greatest force of the concussion is exercised, and along it too, the gases make their exit. The return airway is unquestionably far safer than is the intake airway after an explosion, particularly if solid air-crossings exist in the mine. Many cou'd have made their escape through that way had they been accustomed to this outlet.

The Theory of Coal-dust Explosions.—The circumstances of many explosions, particularly of those on a large scale, cannot be explained fully by reference to gas alone. Those mines in which explosions occur, and are known as fiery, are invariably dry. These dry mines are dusty, containing a large quantity of the mineral charcoal, "mother of coal," which is continually afloat in the atmosphere. The explosions being more frequent in dry mines and deep mines indicate the influence of the coal-dust in extending and aggravating the danger.

Different fine dusts are inflammable and consequently dan-

gerous, according to their degree of fineness and chemical composition. Lycopodium-powder, which is a modern vegetable product resembling mineral charcoal, is highly explosive. Hence it may be a cause, and even the sole cause, of the explosion. It cannot be ignited except by direct contact with an intensely hot flame. There is no probability that the ordinary flame of a lamp has produced explosion from coal-dust alone, nor has it been shown that the ordinary blown-out shot has ignited coal-dust without the presence of some gas. The danger of explosion does exist when both gas and coal-dust are present. It is able to extend indefinitely the transmission of an explosive flame and thus intensify the shock. Experiments have proven that, without any floating dust, the flame from a blown-out shot would not travel more than 25 feet, but that soot would convey the flame to 200 feet The conclus on may, therefore, be drawn that, though coal-dust alone may not be dangerous, in the presence of gas, even in small quantities, it becomes highly so.

A preventative remedy consists in laying the dust by sprinkling. A spray can be delivered into the air-current from pipes 1 inch or 2 inches in diameter under pressure of 50 lbs. per square inch to moisten the air and materially assist in the decomposition of the dust

Ankylostomiasis.—This is a disease attacking miners in the wet collieries, and though but recently discovered in Germany, is receiving considerable attention from other governments because of its infectious character, and energetic measures are being taken to eradicate it Imported into Westphalia from Hungary, in one district alone it has increased from 107 cases in 1896 to 1400 cases in 1902, and during the year 1902, in another district of Germany, 5.29 miners out of every 1000 were afflicted.

The disease resembles cholera or typhoid, the source of contagion being fæcal matter. It is treated by the medical men as dangerous and the patients are quarantined. It is promoted by the sprinkling of the mine with pit-water, and by the high underground temperature. Inasmuch as dry mines of the infected districts do not show so large a number of cases, un-

doubtedly the discontinuance of watering for a time would prevent the development of the ova. The obstacle to this method is the increased danger from explosions from gas or dust. The nature of the disease is not perfectly understood as yet, and many miners have been treated two or three times without the removal of the parasite. Its destruction in human excrement is best effected by the use of fern extract (Extractum Filicis), which proves a better remedy than the thymol used in England. Dry-earth pails, one to every four workers, are distributed to prevent dissemination. The mine owners are aware of the gravity of the situation, and in many districts have discontinued the watering to reduce the plague. That it will in time develop in our American mines, among the foreign element coming from the infected regions, there is little doubt, and careful watch should be kept for its appearance.

Underground Fires.—The most extensive conflagrations in mines are those initiated by gas or powder explosions, communicated subsequently to timbers, canvas, or floating coal-dust. Fires differ from explosions only in the degree of combustion, and either may result in the other. The flame from a miner's lamp or a spark from powder, in contact with any combustible, are the elements producing explosions or fire, and the remedy is to isolate one or all of them. The exercise of care in stables, pump-rooms, and storage-bins for oil, powder, and waste would eliminate a grave danger. In these quarters electric lights can well be provided and the floors sprinkled with clean gravel or sand. In rooms or travelling-ways fires come from defective lamps or exposed lights.

Fires from Blasting Agents.—Powder is not liable to ignite fire-damp because the temperature of its detonation is not over 1000° F. An explosive not properly confined to the hole in which the execution is to be done will be blown out in the room and ignite any combustible gas, if sufficient air be present. Blasting agents which are not chemically perfect may also produce fire and explosion. The unconsumed gases of the detonator are projected into the workings, where they combine with the requisite

amount of oxygen to cause a second blast, under conditions dangerous to life and property.

The employment of a better grade of explosive, the use of a powder free from flame, the ignition of the fuse without spark, the use of water cartridges for extinguishing the flame, the employment of specialists to whom the blast is entrusted, and the substitution of electric firing when the employees have left the works, will lessen the risk.

Spontaneous Combustion.—This is the slow process of burning where heat, combustibles, and the minimum amount of air are the elements involved, and it frequently occurs in a goaf and in abandoned rooms containing broken coal, dust, and rotting timber. Here the roof pressure compacts the material, developing an amount of heat sufficient to partially decompose the ever-present pyrites. This incipient conflagration is furthered by ingress of air. The amount of combustible consumed depends upon the amount of air gaining access to it. Once started, a pressure ensues within the room or goaf, and "breathing" follows, forcing gas out through the trenches in the bottom, at the roof, or from cracks in the side walls and admitting air. At this point the sulphurous fumes give warning, and active measures must be taken to prevent fire or explosion, or both. But a complete isolation from the mine is difficult and a perfectly impermeable enclosure impossible, hence the failures attending the employment of these measures. The safer method of procedure would be the construction of a gas-tight bulkhead, or dam, against the goaves, if they can be perfectly walled. If not, their contents should be removed, though this can be done only at great risk.

Spontaneous combustion can be avoided by keeping cool the combustible accumulations in the rooms or abandoned workings by an active air-current.

Extinguishing Fires.—Besides the remedies already suggested for preventing their inception, the usual method of preventing the spread of fire is to beat out the flames or drown the fire with a flow of water, steam, or carbonic acid. Resort

must be had to the last method when the fire cannot be extinguished by cruder methods, or when it has attained such headway as not to be overcome by simple means. To be effective, however, the heading must be absolutely gas-tight, not only in the bulkheads but also in the upper strata. If they are porous or the mine is shallow, air will enter from above to feed the flame. The mine in that event must be abandoned. A fire at the Calumet and Hecla copper-mines, said to have been communicated to the shaft-timbers by the friction of the hoist-rope rollers, was extinguished by the liberal use of carbonic acid. After the surface had been frozen to stop leaks in the shaft, carbonic acid was injected and the fire brought under control. For the manufacture of 3000 cubic feet of carbonic acid there were used 1200 gallons of sulphuric acid and 4500 lbs. of limestone.

Water is the simplest quencher of flames, provided it can be made to reach all portions which are on fire. But it has happened that into certain rooms the water above the foot of the shaft has not reached the face because of the compression of the air locked in them, which cannot escape. The fire, therefore, raged above the water level until this air was consumed, and in some cases broke out again as soon as the water was removed.

Steam has been injected into some burning rooms for quenching fires, but it has not proven very acceptable.

Protective Measures.—As a prevention against the excitement and confusion arising at the time of an explosion or a fire, the following disciplinary precautions may be exercised before the accident:

1. Consider the quickest and safest mode of descending into the mine when the usual winding arrangements are useless. 2. Plan for the installation of a special winding-engine. 3. Connect water-pipes from the surface to the mine with branches for use in case of fire. 4. Arrange for the fitting of an extra engine and fan for the emergency. 5. Keep the tracings of all working plans to within three months of work, showing all roads then open, the position of overcast, doors, and brattices. 6. Accustom the men periodically to travel certain roads which

they are not in the habit of taking, having finger-boards showing the direction of the upcast pit. 7. Keep on hand all safety apparatus, including the Fleuss machine, a quantity of light air-pipes, and "first-aid" appliances. 8. Appoint, during ordinary working of the collieries, some of the leading officials to act as emergency officers at the time of accident, drilling them in their duties.

Rules for Guidance after Explosions.—The following mode of procedure would facilitate the work of the rescuing parties as presented by Mr. Garforth. Instructions similar to these should be posted about the mine, in various languages, for the guidance of the men and the emergency officers. Each colliery and mine should have a complete map of its workings and a perfect outline indicating correctly the path or paths pursued by the ventilating current in its circuit through the mine. It is necessary when accident occurs as a guide for the rescuing parties, and, although the mining laws do not require this provision, it would be well if maps were quickly available.

1. Send for the emergency officers, assign to each his duty, and appoint some one as deputy in the event of serious accident to the manager. 2. Examine all old connections with the shaft and arrange to repair broken stoppings. Prepare stretchers and stimulants and arrange for a hospital. 3. Provide exploration parties of five with leaders, supplied with safety-lamps, mine-plans, restoratives, cylinders of oxygen, and a stout cord. On no account must any one enter alone, even on the shortest journey. 4. In advancing, the party should move in single file, the leader of each search-party alone testing for gas. Do not let a safety-lamp be the final guide as to the absence of after-damp. 5. Loss of life to explorers may, perhaps, be avoided by remembering the dangers: (*a*) after-damp; (*b*) falls of roofs and sides; (*c*) underground fires and consequent risk of a second explosion. 6. If the force of the explosion has blown out the separation-doors and overcast, they should not be restored because of the possibility of undiscovered fire. 7. Main intake airways, blocked by falls, must not be traversed unless the men carry with them

an unrolled brattice-cloth, which will admit of a double current of air through the airway. The brattice should be non-inflammable. 8. To discover the existence of fire, restore the ventilation and examine the return airways every hour for (*a*) fire-

FIG. 171.—A Diving Knapsack.

stink and (*b*) a rise in temperature. If the former be noticed, that section from which it comes should be explored first, and its fire extinguished if possible, or it should be closed off by stoppings, or, in extreme cases, the pit entirely closed. 9. Parties should be careful not to go too far at once, even when taking air with them, as the force of the explosion will have forced the

after-damp into the interstices of the goaves, whence it will gradually exude.

Aerophores.—For penetrating a very impure atmosphere aerophores of different makes are to be had. They consist of a portable bag or cylinder carrying enough compressed air or oxygen for the respiration of a miner, and his lamp, while making repairs or exploring. The oxygen is inhaled by one tube, while through an exhaler is ejected the CO_2, which is absorbed by caustic soda, leaving the N only to return to the bag. Fleuss' apparatus looks like a knapsack, weighs 28 lbs., contains a four-hours' supply of oxygen, and has besides a self-contained illuminator—an acetylene lamp does not depend upon the oxygen of the air for its burning. A lamp burning methylated spirit heats a plug of lime and renders it incandescent.

The Fleuss diving knapsack (Fig. 171) consists of a cylinder and a cell in four compartments with a perforated false bottom. The cylinder contains oxygen at 240 lbs. pressure, and delivers the gas to the nostrils by a tube. The carbonic acid gas is exhaled by the diver into the cell, where it is absorbed by caustic soda. The entire combination carries a four-hour supply, and has done excellent service to rescuing-parties after accidents arising from fire and inbursts of water or floods of gas.

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PART II.

PRACTICAL MINING.

CHAPTER I.

SHAFTS.

Shafts may be sunk for permanent or temporary purposes, and they may be intended for one especial purpose only—of hoisting, travelling, or ventilation; or their size may be sufficiently large to warrant division into a number of compartments, one for the pumping and ladderway and the remainder for hoists, according to the output. Collieries require additional communication with the surface for ventilation. The large area required for, and the foulness of, the return air demand a separate outlet for the upcast, as also for the intake, which should never be interfered with by hoisting.

The numerous drawbacks to the single-entry compartment shaft or slope are so fully recited, page 20, that only in vein-mines should the development be thus risked. Certainly the ventilating-ways should not be in adjoining compartments, because the bratticing could never be kept tight enough to prevent a leakage of fresh air into the upcast. Only unusually hard rock, or exceptional difficulties in soft or watery ground, warrant a single entry. Where a prospecting drill-hole has been used to test the ground the shaft should not be carried down along on it. It could eventually be of greater service as a ventilator and a ropeway than it could be capable of during sinking.

When it is desired to remove the mineral quickly, several shafts are sunk, their positions being a matter of indifference. Ordinarily, however, the location of a shaft and its equipment is a matter of vital import. The configuration or nature of the surface affecting transportation may govern the selection of a site; but, *cæteris paribus*, the principal shaft should be so located as to reach the lowest point of the workings. This is not at the outset always possible to do so we are accustomed to see one shaft after another abandoned or relegated to secondary uses. Instance the numerous illustrations from the Lake Superior region. The Calumet and Hecla has eight shafts, each over 1000 feet deep, with a complete plant over each one. Nor is this an exceptional case.

How Deep can We Mine?—This question is not now the serious one that it was twenty years ago. The natural limits as determined by physical conditions vary Undoubtedly the maximum depth in the copper-mines of Lake Superior will always be greater than in porphyry districts. The question of ultimate depth would be a limit approachable by the possibility of installing at the surface sufficiently large hoisting-engines and stout enough wire ropes to lift the live load and the weight of the rope itself. These mechanical difficulties have a positive limit beyond which no mining can be carried.

Again, the possible reward of mining compared with the expense of operations would place a limit of depth which is independent of the mechanical question. Judging from the cost of mining at the increasing average depth in copper-mines, it must be granted that there appears to be no fixed relation between depth and running expense. The main expenses of mining, such as stoping, tramming, and superintendence, are to some degree fixed, but when the heat begins to be troublesome with increased depth, the cost of the working is increased. Pumping need not be a serious element in determining the question of cost, for experience shows in the deep mines a proportionately smaller quantity of water, and a lower price per ton of output, than existed during the earlier history of the mines. Increased

depth means increased rock pressure, and necessarily stronger braces in shafts and all openings.

On the other hand, the increasing output and the general improvement in mechanical and engineering appliances have resulted in a saving much greater than the increased cost due to increased depth. Undoubtedly, at the present time, with an estimated depth of mine twice that of fifteen years ago, the total cost of construction is less per ton of output. Again, the increased depth results in increased capacity of the mine, which should be followed by a corresponding increase in the capacity of the shaft and stoping, and by a change in the methods of hoisting. Instead of a single compartment with the intermittent hoist of a single cage, there must be a number of skips or cages in each of several compartments. This condition must limit the depth to 6000 or 7000 feet.

The results of the general discussion indicate that the limit might be placed in the Transvaal or our Michigan copper-mines at about 8000 feet, unless a discovery of exceedingly rich ore-deposits below this depth should warrant exceptional appliances for their recovery.

Some of the deep mines and shafts of the world are mentioned below:

In Austria-Hungary 7 are deeper than 1500 feet; in Belgium 12 exceed 2000 feet in depth; in France 5 are over 2000 feet; Germany has 10 over 2000 feet; in Great Britain are probably 100 exceeding 1500 feet and 20 more than 2000 feet; Norway has but one deep shaft, 1900 feet; in South Africa are 10 exceeding 1200 feet in depth; and in the United States are dozens over 2000 feet deep. Of these, the deepest shafts in the several nations are indicated below:

Adalbert Przibram, Bohemia, Austria-Hungary	3672
Maria Przibram, Bohemia, Austria-Hungary	3360
Produits Colliery, Mons, Belgium	3937
Viviers Shaft, Gilly, Belgium	3750
Montchanin Colliery, Le Creuzot, France	2300

Kaiser William II, Clausthal, Harz, Germany	2900
Einigkeit, Lagau, Saxony, Germany	2620
Pendleton, Manchester, Great Britain	3474
Ashton Moss, Manchester, Great Britain	3360
Robinson Deep, South Africa	1991
Red Jacket, Calumet and Hecla, Lake Superior, U. S. A.	4900
Tamarack, Lake Superior, U. S. A.	4450
Lansell's Bendigo, Victoria	3302

Shafts sunk to facilitate the execution of long tunnels are best located with their axes in the plane of the tunnel, affording better alignment, and only because of the difficulty of supporting the shafts at the tunnel level is it the common practice to place them at the side. Shafts are, however, losing their importance for this work, since the introduction of the rapid, ventilating, drilling-machines.

As regards form, the rectangular is the most common (Fig. 173). Its timbering is easily accomplished, and the best adapted to loose ground. Where brick or stone is used instead of wood for lining, the sides are arched to give great strength, and this perhaps led to the round or elliptical shapes, which are such favorites in Europe on account of their greater resistance, and particularly because of the loose soils and watery strata encountered. That their entire area cannot be utilized is, however, an objection (Fig. 172). The timbering of the polygonal (12 to 16 sides), used in Belgium and the north of France, is not so easy to fit as is that in the hexagonal or octagonal shafts.

The dimensions of the shafts, governed by the number of compartments, should be carefully studied to meet all requirements of strength, output and escapement for a prolonged period. The size increases as the depth and output increase. Outputs of 100 tons were regarded as large not so long ago; but now many hundreds of shafts have a capacity of 1000 tons daily. Colliery shafts are built of a greater area than those in metal-mines, which latter have less traffic, besides being restricted generally by the distance between the walls. The size of the

compartment is determined by that of the bucket, skip, or cage, its length being the width of the shaft, the length of which is governed by the number of divisions (see pages 23 and 203).

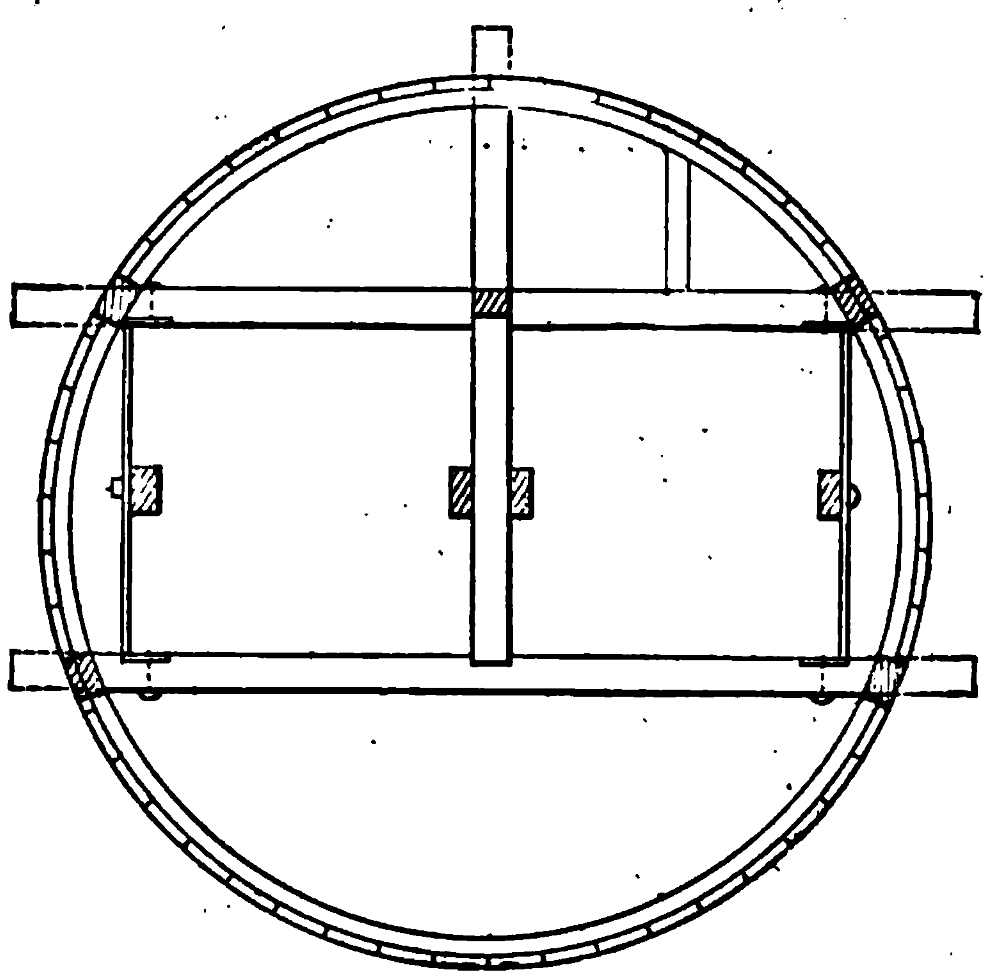


FIG. 172.—Divisions of a Circular Shaft.

Compartments placed side by side make a stronger shape than if arranged in a more compact form (Figs. 173 and 174). The compartments for metalliferous cars are about 4′×5′; those for the coal-cars, from 6 feet to 8 feet wide, by from 10 feet to 12 feet long, measured inside of the timbers. The common sizes for coal-shafts are 10′×38′, and 12 ×24′, with wall-plates of some even 50 feet long. In the Lake Superior iron region the shaft dimensions are about 9 feet long and 20 feet wide. In Montana and Nevada smaller sizes prevail, while in Colorado a single compartment suffices for the small outputs of high-grade mineral. The largest shaft yet begun is a nine-compartment shaft 38′×42′ in the clear. Circular shafts for buckets holding about 1500 lbs. are 8 feet in diameter; for cages 13 feet. The sizes of the ventilating shafts are a matter of in-

FIG. 173.

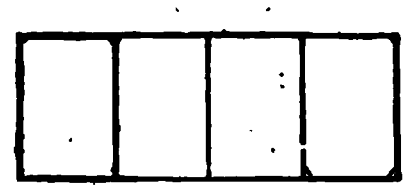
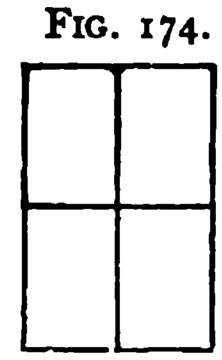


FIG. 174.



The Four-compartment Shaft.

FIG. 175.—The Timbering of a Four-compartment Shaft.

difference, so that they transmit the necessary volume of air with the minimum resistance, and at a current velocity not exceeding 1000 feet per minute. The upcast shaft is therefore usually round, and the downcast a walled rectangular. Neither should be housed, though the former for a furnace ventilator may be provided with a chimney high enough to prevent the distraction of the current by surrounding buildings; or with traps closing tightly and quickly if a fan is used. An area of 1 sq. ft. for every eight men employed is a good basis for the upcast of a moderate-sized mine.

The features governing the selection of a site have already been examined on page 20; so there remains to consider the process of sinking. In a soft-ore lode the shaft section should reach from wall to wall, and massive shaft pillars be maintained, else it is sure to succumb. In hard-rock lodes the shaft should preferably be on the foot-wall; on the hanging-wall heavy supports are necessary, especially if the country rock is porphyry.

Sinking a Shaft.—This process is slow because of the difficulty of putting long angling shot-holes. Small shafts are sunk by hand cheaper than by power-drills, and almost as expeditiously, unless perhaps the continuous system (see Fig. 286) is used; and the loss of time in removing all the implements for each shot bears a large ratio to the total. Even in drifting, the actual drilling heat is not more than half of the whole time. The number of men depends upon the size of the shaft opened; only two miners can drill to advantage on an area of 20 sq. ft. A larger size gives more room proportionately to each miner, and permits faster work, and in a shaft 10′×11′ there is room for three pairs of miners. This space will accommodate two machine-drills, which in ordinary rock can make 5 feet of advance per day (divided into three shifts of eight hours each). A shaft long in proportion to its width, sunk by two or four machines, has two centre-cut ranges of holes (Fig. 309), which are independently fired. The cost of sinking is from $5 to $18 per cubic yard. Below 100 feet the rate increases each 100 feet almost as the square root of the depth. Rziha says that in Europe the

cost of excavating shafts is from 50 to 100 per cent higher in wages, and the cost of putting in timber 15 to 30 per cent higher in wages, than the estimate for the same amount of tunnel-work. In the Lake Superior region one lineal foot of average shaft costs as much as a lineal yard of gangway and a cubic fathom (216 cu. ft.) of stoping.

No figures can be given for calculating the cost of any kind of rockwork. Local conditions of labor and supplies vary too much.

A temporary frame is erected just inside of the permanent posts of the head-gear and over the end of the shaft opposite the pumpway. Here the hoisting machinery is located and the conveniences for loading of buckets. The pumpway is partitioned air-tight, to serve for a ventilating-way as well as to contain the various pipes. Precautions are taken to prevent the material falling down the shaft by providing a fencing around the opening and spanning it by a wide-gauge track upon which a platform car travels to receive the loaded bucket and exchange it for an empty one. Often a hood is provided for the protection of the miners against fallen rocks; also trap-doors at the surface, unless the ventilation is very poor.

The work of excavation is begun within the frame, which latter serves as the template for the remainder of the timbering. At 6 or 8 feet down, a temporary scaffold is erected across one end of the shaft to receive the material thrown upon it, whence it is lifted to the surface. When it has progressed 15 feet below the surface, the first permanent curbing is put into place. Through the first score of feet the progress is quite rapid. Beyond this, progress depends on the windlass or engine. For 90 feet a windlass will suffice, but deeper than this an engine must ultimately be used. Except in small operations the engine is placed at the start. The entire bottom is attacked at once, a small corner sump being carried in advance for drainage and for "bearing in" while shooting.

Extending Shafts by Pentice.—Shafts are prolonged without interference to the regular mining operations, and without

danger to the shaftmen, by opening only that portion of the shaft area not under the hoistway for a distance of 12 or 15 feet, and then widening it out to the entire size of the main shaft. This

FIG. 176.—A Pentice.

leaves a roof of rock ("pentice") (Fig. 176), that shields the men. When another lift has been started the pentice is cut away, and another started for the next drop.

The Service of Shaft-timbers.—There is no safety nor economy in the practice of leaving a shaft untimbered, even if the two walls are hard and self-sustaining and can be dressed smooth. There is a thrust from the walls in the country rock which results in the release of fragments that may injure the men by their fall; a tendency to movement exists which may completely close the openings, and the vibration produced by rapid hoisting tends to loosen material; hence timbering will be required, to furnish a close lining to the shaft and a rigid support to the cage-guides. The timbers, therefore, are to serve mainly as a means of preventing movement in the walls, which, having once begun, can be checked.

The Character of the Timbering of Shafts, whether vertical or inclined, depends upon the speed desired for hoisting. It is more substantial for cage-hoisting, or mechanical haulage, than it would be for a bucket-hoist or animal haulage. The varying conditions as to the character of the rock, the area of the shaft, and the firmness of the enclosing strata, determine the choice of system to be employed. An opening, for the shaft made any greater than is necessary to accomplish the desired end, not only increases the expense of opening, but also the cost of subsequent maintenance The timbers become needlessly long; the cost of breaking, high; and the material to be transported, larger than necessary. Moreover, the cost of back-filling the useless space would be materially increased and the size of the timbers in a horizontal direction must be proportionally larger for a given rate of pressure.

Timbering a Shaft. — The timbering of a shaft may be done simultaneously with or subsequent to the process of sinking, according to the firmness of the ground through which the work is carried. Through loose ground, or for the first 30 or 40 feet of depth through the alluvium, the timbering is placed as promptly as possible and of a thickness, compactness, and strength depending upon the nature of the ground, whether loose and rocky or wet and sandy. This continues until hard rock is reached, after which some form of timber-frame is introduced. If any

water is encountered, water-rings or coffer-dams are built after the usual plan (Fig. 177). At the line of union between the

Fig. 177.—A Clay-puddled Shaft-crib.

Fig. 178.—The Timber-frame of a Shaft.

alluvium and the bed-rock a water-tight joint is made. The shelf of rock is dressed level at a sufficient distance below the upper surface of the stratum to allow for the packing, and on it

is built a wedging curb of segments of heavy timbers. This curb is tightly wedged and packed with puddled clay.

The standard frame consists of four pieces (Fig. 178), each timber of the set being known as the plate. At the sides, *RR*, are wall-plates, and the short plates at the ends are end-plates. If the shaft is to be divided into compartments it is done by the use of buntons, *BB*, or girts bolted to or gained into the wall-plate. The greater the length of the wall-plates the stouter are

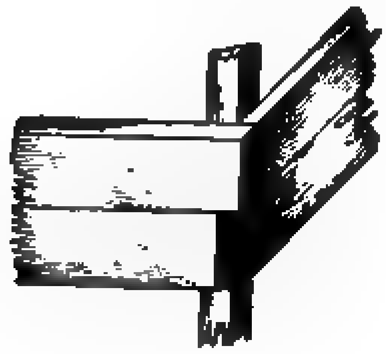


FIG. 179.—Boxed Joint.

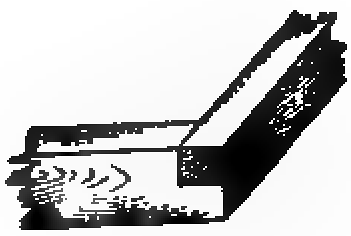


FIG. 180.—Boxed Shoulder.

FIG. 181.—The Boxed-shoulder Crib.

the buntons or girts. On the inside of the end-plates and the girts are spiked the guides for the hoisting-cages. If any compartment is to serve as the travelling- or the pumping-way, or even for ventilation, it is partitioned off by planks nearly vertical to branches, as in the right-hand compartment (Fig. 175).

Timber Joints.—The joints in the frames are of several forms, regard being had to the expense of framing and the convenience of handling the pieces as well as the efficiency of the joint for resisting pressure The timbers may be rough-hewn, particularly if solid cribbing work is to be erected, but preferably with their lengths dressed to a particular template. In firm non-decom-

posing ground the timbers experience little pressure, and therefore stability rather than strength is sought, the latter being secured by ample shaft pillars. Under such circumstances a line of stiff guide-planks will be sufficient, placed in close contact, or in a frame, according to the liability of the material from walls of the rock to scale off and fall into the shaftway.

The simplest form of joint for such a lining consists of 2-inch planks cut to a square butt-joint at the ends to a correct length and placed with their longer dimensions vertical, so that the alternating sets break joints (Fig. 179); in other words, the wall-plates of one set overlap and rest upon the end-plates of the set below, and in turn support the ~~ends of~~ the end-plates of the set above. The end-plates are correspondingly cut. It is efficient and cheap for curbing shallow shafts. It is finished by spiking firmly from top to bottom four triangular corner-pieces.

Shaft-linings.—In Figs. 180 and 181 is illustrated a form of curbing adopted for somewhat heavier pressure, in which the four pieces in the set are cut to the square-box joint to exactly match. These timbers are 4 inches wide by 6 or 8 inches high. In Fig. 182 is illustrated also a somewhat similar system for slightly heavier timbers, in which the framing of the boxed shoulders of the four pieces is vertical instead of horizontal. If these are cut to template, it is not necessary that the timbers should match in height for each set. Corner-pieces are used inside of this frame.

This form of casing is built in sections of 30 feet each in height by placing a pair of 10-inch stulls in a securely horizontal position and building up from them the sets as indicated. During the progress of this timbering the corners are plumbed continually, and waste rock is packed closely with its upward progress. Two men can complete one section of a shaft 5 feet by 9 feet in dimensions in four days' time with one or two helpers at the packing. The men are supported on a cradle suspended by a rope from the upper stulls.

Two-compartment shafts can also be lined by this form of casing, when buckets are used for hoisting, but for cage use the

timbers are not secure. Cribbing is used for small shafts, as it is cheap and simple, but would be too expensive and cumbersome in shafts of two or three compartments.

It is not necessary that these timbers should be in close contact, but the framing may be made such that a slight open space is left between each set of timbers by leaving the tenons larger than is necessary for the shoulders on the ends.

A more efficient joint is shown in Fig. 183, when the pressure from the wall-rock is great enough to require the lining of con-

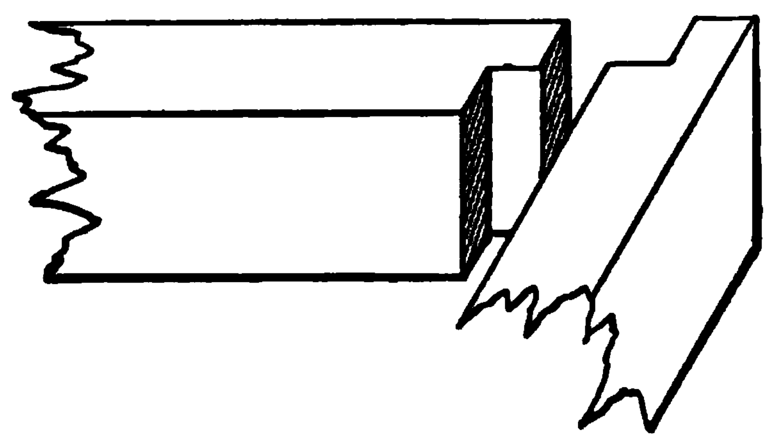


FIG. 182.—The Halved Joint.

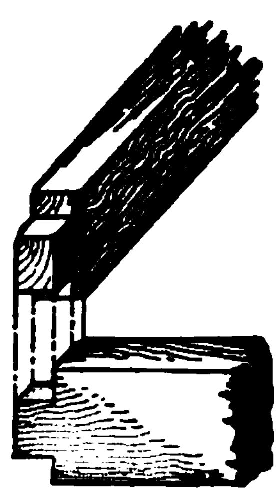


FIG. 183.—The Bevel Hitch.

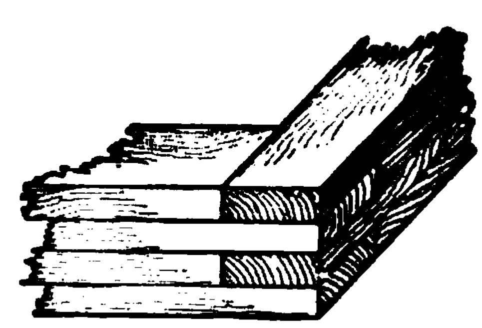


FIG. 184.—The Boxed Joint.

siderable thickness. In this casing a half-shoulder is cut at each end to a true fit, and the pieces laid for smaller timbers. The bevel hitch at the ends weakens the timber unless its full strength be obtained by bringing it to a close bearing with the bevel face of its mate.

This character of casing may be employed when passing through dry quicksand or running strata, in which case the planks would be spiked upward from below as fast as they were inserted. This curbing is arranged with the layers in

alternate thicknesses, measured in a horizontal direction, so as to present the uneven surface at the back of the lining against and upon which the sand may press. For example, the

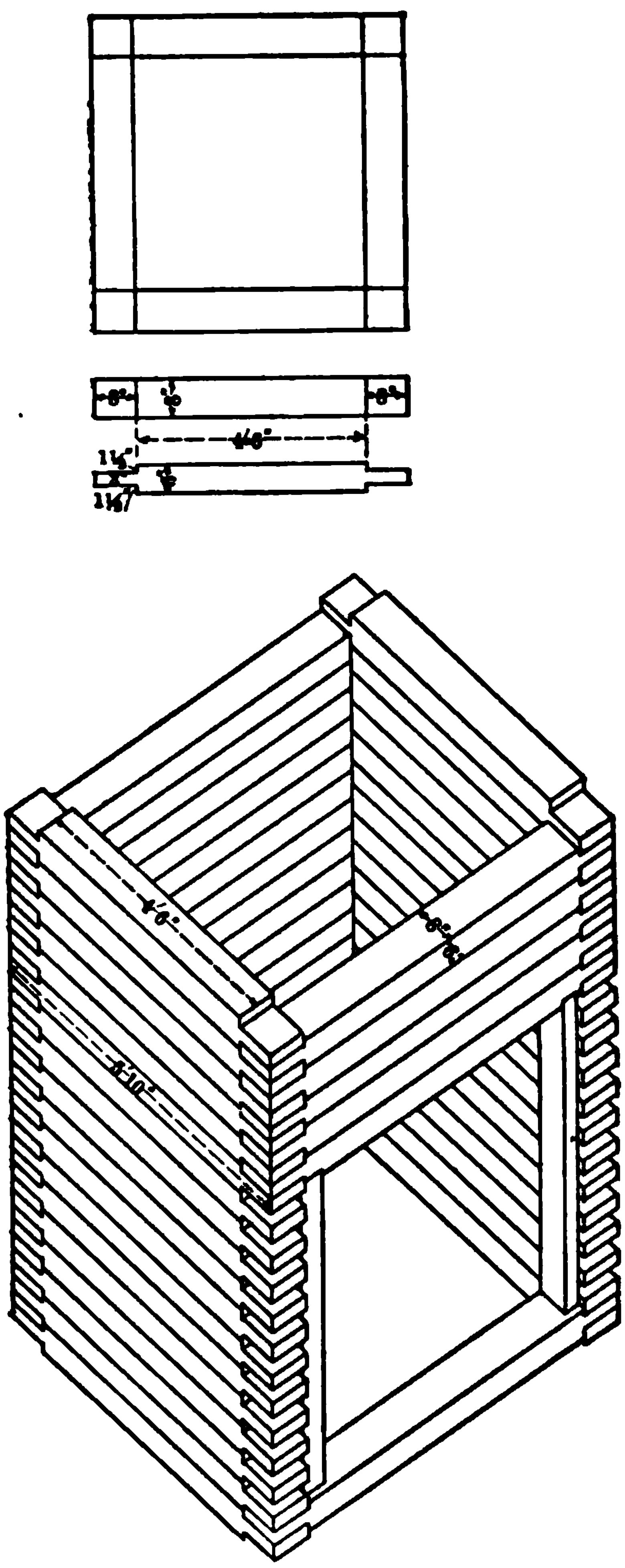


Fig. 185.—Shaft-timbering at a Landing.

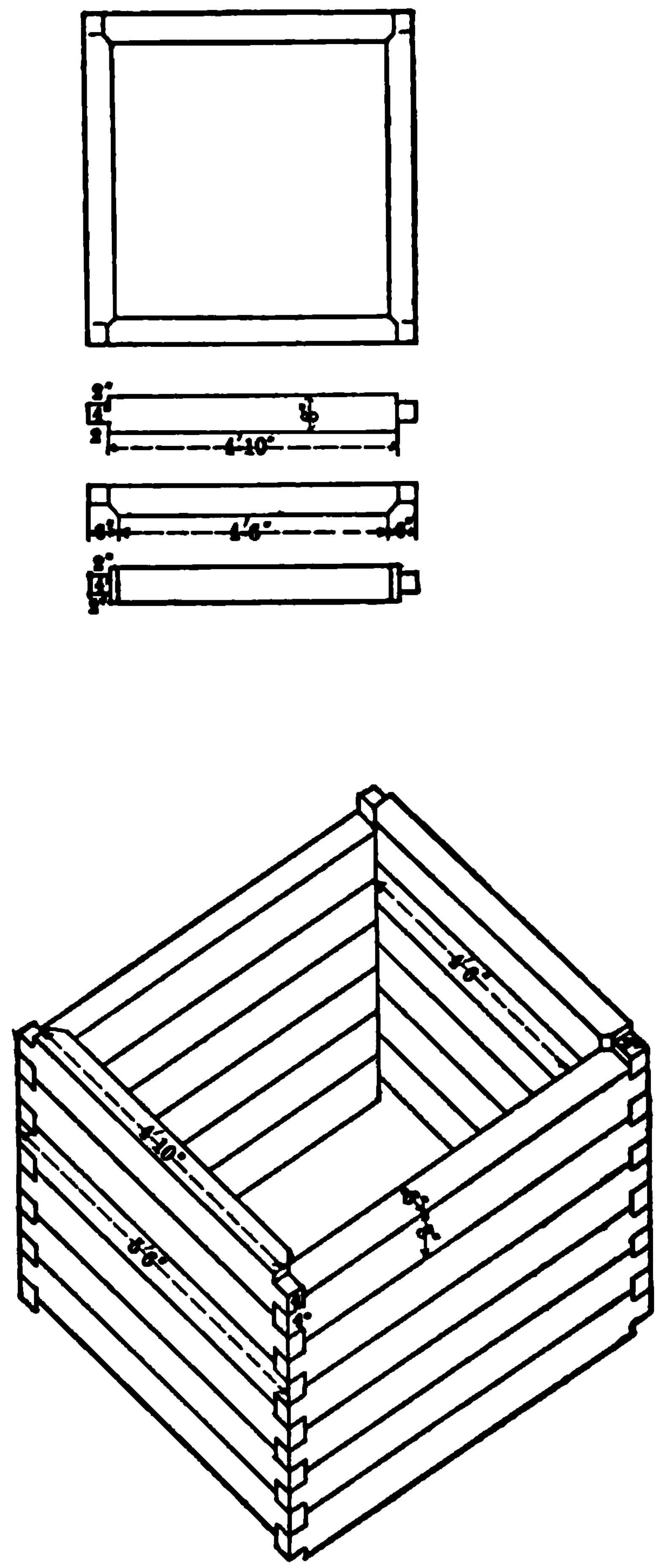


Fig. 186.—Solid Shaft-lining with Bevel Hitch.

planks may be in sets 2 inches high and 8 inches thick all around, alternating with planks 2 inches high and 10 inches thick.

In exceptional cases this form of timbering may stand safely without corner-pieces, but it is wiser to employ them, utilizing also waste packing to keep the joints in position. This may be also arranged in such a manner as to leave intervals between the sets through which some material can be removed to relieve the unusual pressure.

Another method of open framing of wall-plates resting in the end-pieces, much like the log-cabin style, but without any notches at the ends of the pieces is in frequent use in firm ground.

Where the strata are self-supporting, the planks need not be more than 4 inches thick, set on the edge for a depth of 150 feet, or 3 inches thick for a depth of 120 feet; but if the ground is soft, or crumbles, or there is a flow of water, other provisions must be made to increase the strength of the set or to provide a supplementary dam to relieve it.

It is difficult to determine the requisite size of the timbers in a shaft, for the amount of pressure from the walls cannot be measured. Experience with the character of the rock, its strength and freedom from cleavage planes, can alone furnish any guide to the engineer. The dimensions of the timbers are therefore fixed by other conditions, allowance being made for emergencies.

Square-set Timbering.—Shafts are more commonly framed in square sets (Fig. 187). Square-set timbers for shafts are 8 inches square and consist of side wall-plates, end-plates, and four posts supporting the horizontal frame. A set consists of four plates and as many girts, buntons, or inner struts as there are hoist compartments. A three-compartment shaft is shown in Fig. 188 with lagging for lining. The plate-joints are halved with a hitch, or square shoulder-notch, 1 inch deep, cut into the tenon for a support to the post. A centre line is marked on the inside face of the plates, from which all measurements are taken for tenons, mortises, and mitres, as well as for plumbing.

The timbers are prepared by boring four 1-inch holes, by a tem-

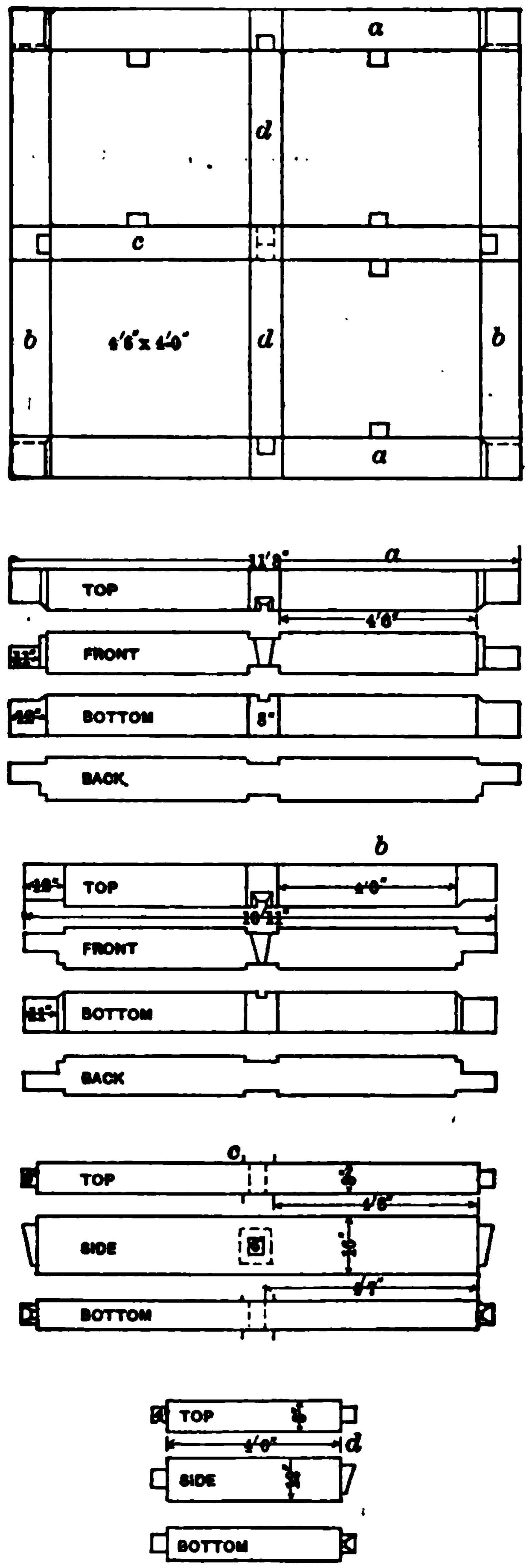


FIG. 187.—A Four-compartment Square Shaft.

plate, near the ends of the wall-plates. Through these are forced iron rods with a long screw-thread at one end and a hook at the other. The length of these rods is something more than the half distance, out to out vertically, of the frames. Some of

FIG. 188.—A Lagged Lining.

the rods have a ring at one end instead of a hook, and other rods in pairs are made of a length of the odd multiples of the half distance. A set of liberal-sized washers and nuts is supplied for each pair of wall-plates.

Setting the Timbers.—If the shaft is started on the hillside, affording height for the convenient disposal of the débris, a supporting frame, from which is to be suspended the first section of timbers, is laid on the surface. If started on level ground, where height must be obtained artificially, a top collar is laid on the timbering above the surface, which timbers are heavy, and in a frame of four pieces with the shaft-opening in the middle. This frame is then reinforced by an outer crib, between which is a filling of rock. The whole structure must be blocked firmly. If the ground is loose, the side-plates are extended far beyond

the limits of the shaft and bolted to heavy cross-sills for support till the firm ground has been reached.

The upper set having been placed level with two of the suspension-rods in each wall-plate with hooks down, two wall-plates of the next lower set are attached with their hooks up. The end-plates are then laid on the wall-plates, the posts are set in their hitches, and the nuts are screwed down tightly. The posts are plumbed, blocks inserted, and wedges driven. Spiked to the back of each plate is a 2″×4″ strip, on which rests the lagging or lining. After the filling has been stowed the process is repeated for the next lower set.

Two-compartment shafts are made in two divisions of the same size, framed identically alike, the longer wall-plates being strengthened by girts and centre-posts.

In the three-compartment shafts the one assigned to pumping is larger than those intended for hoisting, and its girts and posts are more rigid than at the other end. The process is the same as for the single-compartment, the buntons being fitted into the mortises in the wall-plates after the wall and end-pieces have been laid. The wall-plates should be in one piece unless inconvenient. But if in two pieces, the short one should be at the pumpway, with the divisional girt located at the joint. The eight posts are placed, bound, blocked, wedged, and lined up.

The four-compartment shafts with three hoistways are in single line, the fourth being used for lowering timbers and machinery, and the framing is identical with that for three compartments. Those constructed with the compartments in two rows (Fig. 174) have their end-plates longer to enclose a double width (Fig. 187). In this event the stations are in the corners and the direction of the shaft-frame is determined by the subterranean requirements. The excavation is square, the timbering compact and rigid and not excessively long, the shaft has greater capacity as compared with the straight line, and there is but 5 cu. ft. more of ground removed per foot of shaft than in the latter. The middle framing in the direction of the hoist is often braced with diagonal struts to solidify the sets.

Timbering in Firm Ground.—If the rock is firm enough to admit of sinking 50 feet or so before commencing the timbering, the process can be more cheaply conducted by building upward from reachers or stulls set into both walls. On these, four pieces are framed to the studdles or struts at the corners and at the compartment partitions. On these struts a similar set is framed 6 feet above, to support another parallelopiped, and so on up. Planks ("lagging") are driven in around these frames, and the spaces to the rock filled with broken waste. The joints of each timber are of the pattern shown in Figs. 189 and

FIG. 189.

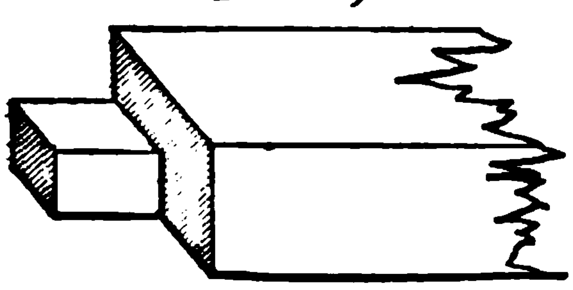
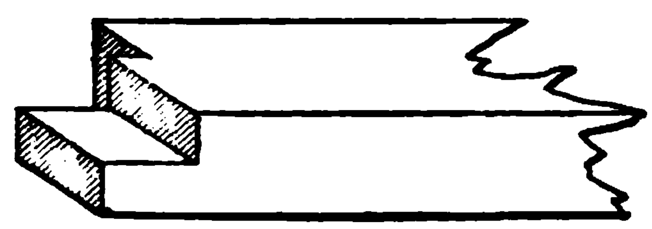


FIG. 190.



Joints of Square-set Timbers.

190. The end-plates and struts are usually 8 inches square, while the wall-plates are laid 8 inches vertically and 10 or 12 inches horizontally. Fig. 177 illustrates another form of timbering rectangular shafts with vertical corner-plates and horizontal lagging.

Shafts such as the Comstock, 6'×24', for continuous heavy hoisting, are fitted with timber as much as 14 inches square, and lagged with 3-inch plank. Where friable rock is penetrated, the frames are braced by inclined struts that prevent settlement. When the ground is friable, marly, or wet, the methods assume a caisson character. Another plan comprises a stout framing (*C*, Fig. 177), inside of which is another strong planked cribbing, between which clay is puddled to exclude the water. The B. & O. shaft at Taylorsville, Ind., was thus successfully carried through quicksand; the outside crib was of 12-inch, the inside of 10-inch timbers, with a 4-inch puddled wall. The famous Hollenback shaft, 45' 4"×11' 6" inside, has a 12-inch clay wall for 31 feet of depth (Fig. 191). It was designed for a daily output of 2500 tons of coal.

FIG. 191.—Timbering of the Hollenback Shaft.

Repairing Timbers. — If the timber shows signs of giving way, other means of securing the shaft must be invoked. With expert timbermen the joints may be strengthened or the frame replaced, but it is preferable to reinforce them by intermediate sets.

After supporting the several sets on either side of those to be repaired, the faulty pieces can be removed one by one, excavating enough ground from behind each post to allow of its being driven back from the shaft clear of the timbers and then replacing each piece singly, wedging and driving it into place from behind. It may be possible also to chop away the piece if it is not desirable to open up more ground behind the set. The new post is wedged into position before the next post is inserted. When plates are to be removed and new ones inserted, the posts, the lagging, and the blocks supporting that side must be removed and one piece inserted at a time. If it bends with the ground and will not stand during the pressure, false timbers must be used to hold the walls in place.

Building Landing-stations.—The stations are prepared at various depths of about 100 feet each, from which the levels are run into inner workings of the mine. These stations must be in excavations large enough to serve as centres of distribution of cars into the rooms and of storage of the loaded cars preliminary to hoisting. Their height is necessarily greater than in the level, and their width is at least that of the longer dimension of the shaft. The timbering and lining of such working stations is a component part of the shaft system, with such modifications as are necessary to connect it with the balance of the works, besides affording a full, clear, unobstructed width for free movement of cars and men. The timbers of the stations usually consist of the four-piece level set, the caps, however, being very heavy, frequently of iron. The lagging at the roof is of 3-inch plank. A height equal to that of two sets of shaft-timbers will afford ample height for handling timbers, pump-pipe lengths, etc.

In opening a station, four-hitch timbers are wedged under the last set of the shaft crosswise, and the shaft continued some distance below. A three-post cap and sill-set is placed against

the shaft-timbers and finished. Portions of the wall-plates lying between the shaft and the station, breast high, are then sawed out, after which the timbers may be continued backward, slanting downward as far as required to usual dimensions of the drift (Fig. 185).

For inclines the stations are of two kinds: one arranged so that the ore-cars will dump directly into the hoisting-skip; the other is a large ore-bin built below the station at one side, from which a discharge can be made into the skip at proper intervals. The latter is far more economical in the long run. The skip-loading, moreover, is independent of the haulage system. But the excavation must be larger than for the former.

The sets are lined up by straight-edge and plumb-bobs, the

FIG. 192.—Station-cribbing on a Slope.

former being long enough to overreach the distance between two sets. A frame can be built in such a manner that a plumb-bob will swing free within small limits from the upper end of the straight-edge, the lower member resting upon the bottom sill.

In Fig. 192 is shown a station built in cribwork.

If the expense would warrant it and the diminished area is not objected to, a second lining might be inserted and prove secure. In the Lake Superior region mine caissons were in-

voked after various futile experiments had been made with other methods of reinforcing the shaft-timbers. For example, iron caissons were forced down inside, or outside, of the old timbers, according as the ground was stiff or soft. In the latter it was even possible to drive them by their own weight, with very little supplementary power. These cylinders, in segments and sections, were added one by one at the surface, keeping pace with the progress of sinking. In one instance a cast-iron cylinder 15 feet in diameter and an inch thick was forced down at a rate of 2 feet per hour through 80 feet of morainal matter

Forepoling.—In very loose ground the timbering must be conducted promptly by some method of shaft-lining. The usual plan is that of spiling (Fig. 193). A set of planks is driven nearly vertically, blocked and wedged into position. Each piece bears against the walls and is fixed as the portion of a complete lining to the entire shaft. In soft or running ground which cannot be tim-

FIG. 193.—The Forepoling of a Shaft.

bered or requires timbers excessively heavy for lining, the method consists in holding back the dangerous ground below the last set in position by an enclosing and protecting shell of forepoling, as described in Chapter III for levels (Fig. 271). The forepole is a stout plank sharpened at its foot, often shod with iron, and has a length greater than that of the distance across two sets. It is driven by a sledge, a single plank at a time, from the back of the plates in the last and lowest set. Little by little the material is removed to allow of it being driven a short distance. Each

plank around the shaft is driven in turn until the entire shell has been advanced. This process is repeated until a new set can be inserted, behind which and in front of the previous lining of four poles is driven another set in a similar manner. The forepoling is started at a considerable angle outward, but in its downward progress is forced toward the vertical until, by the time an advance has been made for another set to be placed, the planks are nearly in their desired position, giving security to the frame and affording safety to the men. When the second set of spiles has been started those constituting the first set are driven their full length.

Forepoling has proven very successful, though it requires considerable timbering. It is a perfectly safe method and can be

FIG. 194.—Forepoling.

employed in almost any variety of dangerous ground except quicksand. Frequently, when the ground proves treacherous, there is perhaps too rapid a rise of the bottom of the shaft. In this event the simplest plan is to brace the bottom sets and floor the foot of the shaft over its entire area, leaving only a small opening, advanced first by forepoling (Fig. 194), and subse-

quently enlarged to the full dimensions of the shaft. This plan resembles somewhat that of the Anderson pilot tube, as illustrated in Chapter IV.

Wood Tubbing for Circular Shafts.—For circular shafts the false work descends with the shaft, but the cribbing is built upward in sections from stout reachers of timber, bedded whenever suitable foundation offers, or from a properly dressed ledge of the rock, and firmly wedged against the sides. The timbers, assuming the character of voussoirs, hooped with iron, may be mere short blocks or wedges, or they may be longer timbers forming a polygon with less number of sides. In the latter case they are united by iron dogs. In ordinary ground the sets are held apart by vertical props with a solid packing behind them. The solid walling may also be suspended from a heavy frame at the surface by iron rods (Fig. 195). In any event the joints and fitting receive the greatest care, and many of the old shafts are high types of the carpenter's art.

The increasing scarcity and cost of large timbers, the expense of fitting and maintenance, their short life, and, finally, the corrosion of spikes and splice-plates, with the consequent leakages, have caused the abandonment of wood tubbing, and the adoption of iron and masonry for all permanent ways. The effect of the hot atmosphere of the mine upon timbers is a decomposition that is not always detected on the surface, but once begun, only better ventilation can delay ultimate destruction. Dry timbers should be frequently probed; alternations of wet and dry are exceedingly destructive; wet timber will last longer than dry. Preservatives have been attempted, with much success. In salt-mines steeping in brine gives great endurance. The sulphates and chlorides of zinc have proven excellent antiseptics; and a grand opening offers to the discoverer of a means of freeing the lead ores of the Western States of the obnoxious zinc, and at the same time utilizing it as a preservative.

Masonry Walling of Shafts.—The use of masonry involves but one disadvantage: it presupposes ground that will stand safely for a couple of weeks without much support. Before the

permanent structure can be introduced, a considerable depth must be reached, to obtain a sure foundation upon reachers, or upon a ledge from which the masonry is erected, the temporary timbering and bracing being gradually removed as the construc-

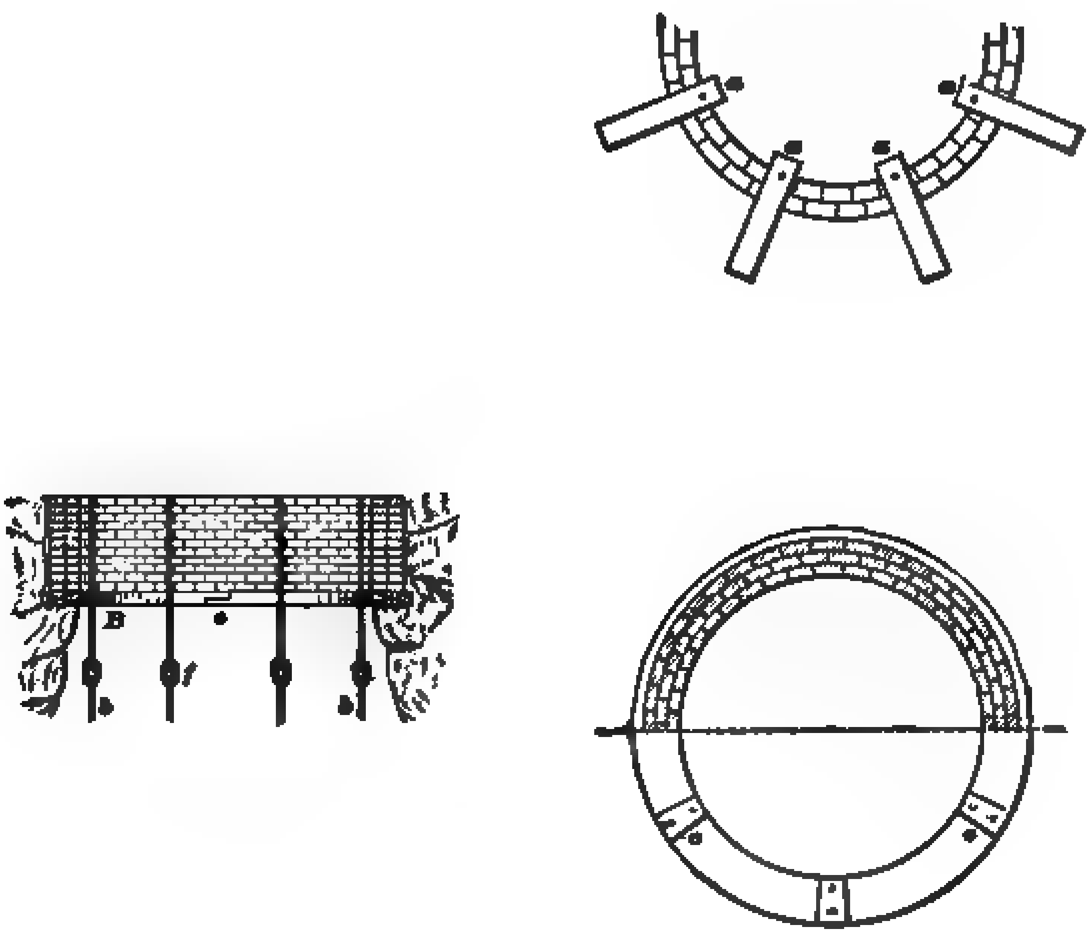


FIG. 195.—Building the Shaft Walling.

tion proceeds. When a very secure ledge or base cannot be had, a wedge-shaped chamber is built for some distance back into the rock from which the solid crib supports the walling.

If the pressure from the walls is not very great, walls are built up of brick or rock, packed securely behind by waste. Often

the mine-water carries matter in solution that cements the whole into one solid mass. When great pressure is expected, the sides are arched toward it; and in very bad ground all four sides are curved, or the circular form is adopted. The arc should be such that its chord is perpendicular to the direction of pressure. In such event, the foundations for the sections are curbs of overlapping timbers patterned to the curve, or of late years of cast iron, with slabs of wood at the joints. The packing behind is carried up with the brick or masonry until the ledge of the upper section is reached, when it is removed gradually and the two sections united. In some instances the masonry compartments are built at the surface and lowered into place. Brick is well adapted for quick archwork. The wall of a shaft 13 feet in diameter is four half-bricks thick; the labor of laying it from a staging is one and one-half days per M. The masonry is supported by rods, *b* (Fig. 195), from beams *a*, *a*, buried firmly in the walls.

Masonry is heavy to support, and not any cheaper now than iron, with which many shafts are successfully curbed. Rings of I beams or channel-bars form the curbs, upheld at proper distances apart, by struts of wood or iron, and backed by heavy planks or $\frac{3}{16}$ sheeting (Fig. 195). English engineers use old railroad iron similarly. Prepared at the surface, the curbs may be lowered into place and quickly set, with little labor. A packing of concrete is used at Saarbruck, giving increased strength and durability. It is estimated that the initial cost of iron lining in place is twice that of wood and equal that of masonry, but the cost of maintenance is one third that of wood and nearly the same as with masonry, if the shaft is dry.

Shaft-bottoms.—A very important feature in shaft-mines is the arrangement at the bottom of the shaft for receiving and disposing of the cars. The loaded and the empty cars should go on and off at opposite sides of the cage, as on the surface. Hence the heading and the position of the engine will be parallel. As on the surface also, the grades to the cage and from it should

be downward. The loading can then be done expeditiously and economically, for not only is the cost of handling less, but the capacity of the shaft is increased thereby. By this means the work of caging can be performed by fewer men than if the tracks were on a level or up grade for the empties. The latter are run down to some convenient point where they are picked up by the locomotive or mules.

If it happens that, in order to accomplish this, the shaft-bottom must be lowered and the main-entry bottom also taken up, the subsequent saving will warrant the initial expense. Occasionally it is found desirable to build a special roadway, passing the sides of the cage, to return the empties to their appropriate entries.

The headings at the shaft-bottom are usually arched with brick, with considerable packing back of the arch, and further protection of longitudinal layers of 4-inch plank. Care should be taken that the space between the rock and the back of the arch be completely filled to avoid movement or side pressure of the roof. The supporting walls to the springing line should not be over 4 feet high, and the curve of the arch as low as will give sufficient height for the work to be done at the shaft. Not infrequently an inverted arch is built for the floor.

Shaft Pillars.—The dimensions of the shaft pillars should be such as to ensure absolute security to the works. Though many local conditions affect their size, the following rule may be given for flat seams, where the depth does not exceed 1000 feet. With D to represent the depth of the shaft, t the thickness of the seam, and R the radius of the pillar, all dimensions being in feet, then

$$R = 3\sqrt{Dt}.$$

In inclined seams it is evident that the larger portion of the pillar should be to the rise rather than to the dip. Its length then is greater than the dip portion by three fourths of d, in which d is the distance in feet along the seam from the foot of

the shaft to the foot of a perpendicular drawn from the top of the shaft to the seam. Then, if R is the length of the pillar to the dip,

$$R=60+0.05\,D\sqrt{t}.$$

The line at right angles to the dip of the seam is the ultimate breaking line.

In a seam 6 feet thick, with an inclination of one in three, in a shaft 1200 feet deep, the dimensions of the dip pillar will be 207 feet. The value for d being 346, the length of the pillar to the rise will be 406.5 feet. The total length of the pillar becomes then 673.5 feet and its width for one shaft twice that of the dip pillar, or 414 feet; for two shafts, 180 feet apart, 594 feet.

In the case of a fresh opening in another seam of coal above or below the bottom already existing, care should be taken that the headings in both seams be not placed immediately above one another and parallel, unless the distance is very great.

The underground s ables should be in a position as convenient as possible, and well ventilated with splits in each separate stable for 24 animals. The archway overhead should be 12 feet by 16 feet.

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CHAPTER II.

SINKING IN RUNNING GROUND.

Excluding Watery Strata.—In he search for minerals, mining is conducted to greater depths and into more treacherous ground, as time advances, and the highest type of engineering skill is called into play as water-bearing strata are encountered. Not only is the shaft to be sunk through them, but their underground currents must be excluded from the mine by water-tight casing, otherwise an elaborate sy tem of pumps must be continually maintained during sinking and mining. Not infrequently the expense of removing seepage-water while sinking becomes so large an item of dead-work as to compel the abandonment of the works. A system of tubbing is, under those circumstances, advisable.

Two varieties of cases present themselves—one in which the ground penetrated is quite firm but porous, and the other includes running soil, marl, quicksand, etc. In the first case, the pumping facilities must be ample, or the water kept back during mining; in the second, the possibilities of excavation are greater than the facilities for its hoisting, and the laborers are in danger of being overwhelmed with soil. In either case the ground traversed must be insulated by a tubbing, hermetically sealed above as well as below the soft measures.

This not only renders sinking possible, but it excludes water and silt from the mine, and permanently dispenses with much of the pumping arrangements. It very seldom comes into play in vein-mines, where, with the verticality of the lodes, water cannot be prevented from percolating into the mine. It is in

the stratified regions that the use of the crib is of the highest importance. Beds of gravel, sand, and clay, or porous strata, percolating large quantities of water, are not easily traversed or held up without a strong water-tight lining, for the pressure of the moving material tends to make the bottom rise, as well as threaten the sides. A deep shaft in such a region may encounter several occasions for such tubs, which under suitable conditions may be introduced in lengths as required, and only to the extent of the soft ground. Still, it would give more substantiality to the work to form one continuous length of tubbing, even across the good ground. It is not uncommon to find in Germany shafts with three sections of iron tubbing, united by lengths of brick or wood lining.

Tubbing.—This process consists in confining the seepage area to the bottom of the shaft only, by building a water-tight cylinder lining to the shaft, and carrying it down with the sinking beyond the wet stratum. In England, a bed of sand called the lower red sandstone, which is almost fluid, has several shafts tubbed through it. In Britain, Belgium, and the North of France, several mines are reached by tubbing through the fissured chalks and marls of the Cretaceous. The Thonmergel of Germany is frequently tubbed to the Bohn Erz, below, dry enough for work. While sinking the Murton pits 4000 gallons were pumped per minute, and the "come in" of water for the Exhall shaft was 1650 gallons. Still, the inefficiency of this plan, sometimes called the English system, is recognized, and several methods better applicable to loose and watery beds have been applied with more or less success. Excepting the Poetsch method of freezing the ground to be penetrated, they are modifications of the diving-bell, or pneumatic pile.

Wrought-iron tubes in segments are bolted on at the surface as fast as the lowering proceeds, until the secure, impermeable bed is reached. Here a smooth base is prepared for one or more wedged curbings, behind which moss or concrete is rammed. The tubbing is backed with rock or concrete all the way up, and connected with the next upper section. The holes in the seg-

ments, for convenience in handling, and to relieve the tubbing of pressure till the work is completed, are plugged up. The early practice of bolting the segments together through the inside flanges was soon abandoned, and now the flanges are outside, wedging, pressure, and friction keeping them. On account of the curious accidents occurring from the pressure of air locked behind the tubes, it is advisable to lay a pipe to the surface for the gas to escape, and, similarly, to relieve the water. A shaft of 16 feet diameter was sunk at a monthly average of 104 feet with four shifts of 8 and 10 men each. Several Canadian salt-mines, having shafts 10 feet 6 inches diameter and reached at a depth of 1150 feet, are tubbed through 260 feet of water-bearing strata, in sections 2 feet high and $\frac{5}{8}$ inch to $1\frac{3}{8}$ inch thick. The columns rested on iron curbs with firm base. The joints are calked.

Masonry Curbing.—This is a much cheaper and more efficient wall in shafts than is iron tubing. With good hoisting machinery, three masons in four-hour shifts finish 10 feet per day of a 16-foot shaft. Towers of masonry, resting on an iron curb with a cutting edge, were built on at the surface; while, to facilitate the sinking, digging was being carried on below, or, if the material was wet, a process of "bagging" was employed. When abundant in size and quality, wood gives great satisfaction, being elastic, easily laid and repaired—qualifications not possessed by masonry. Iron offers the advantages of strength, combined with a facility of handling, which recommend it for large shafts and enormous flow. Though it is not possible to presage or measure the pressure, and thus determine the kind and strength of tubs, a thickness of 12-inch wood, 7-inch masonry, and $\frac{7}{16}$-inch iron may be suggested as common. As a matter of fact, the tubbing should taper off toward the top. In many cases, however, the use of 12 inch staves hooped with iron did not prove adequate, nor did the backing of 12 inches more of concrete help matters; where the shafts were not abandoned, $\frac{5}{16}$-inch sheeting met the emergency.

Sinking through Running Ground.—When a bed is encoun-

tered of a material so soft as to behave like a fluid and be pumpable, the area which can be opened with safety is very small indeed. In this event some variation of the "spilling" processes, a pneumatic pile or some process of congelation, should be employed.

Triger's Method by Pneumatic Tube.—M. Triger employed, in 1839, the principle of the pneumatic pile, in which an iron tubbing was built down to an air-lock which communicated with a diving-bell at the bottom. The atmosphere of the caisson was maintained at a pressure of not over 60 lbs. per square inch, and checked the influx of the sand, which the miner shovelled to a sump, whence it was aspirated to the surface. Meanwhile the tubbing was being forced down from the surface as rapidly as possible. The tubbing was divided into three chambers by partitions with hinged doors, the middle one constituting an air-lock. The natural limitations of this process are the outside earth friction and the physiological difficulties of men working in compressed air.

The Kind and Chaudron Process of Boring Shafts.—In France and Germany the loose, watery marls presented difficulties which the methods described failed to overcome. What with pebbles and fine rock interfering with aspiration, water completely inundating the shafts, and the difficulty in establishing water-tight joints, the operators were routed. In 1850 Herr Kind devised a scheme for mechanically sinking shafts, just as one does a bore-hole, and still further conquered difficulties hitherto insurmountable by a variation in the mode of lowering the tubbing, and by a device for regulating the influx of water. When M. Chaudron added the sliding bottom-piece to form a perfect joint, after the Kind boring-tool had prepared the base, the acme of shaft-sinking was reached. Since 1862, when the first shaft was sunk, 6 feet wide, 480 feet deep, at a cost of $450 per foot, not a single fatal accident is recorded against the process, which owes much of its success to the fact that the sinking and lining are completed before a man enters the shaft. Two abandoned shafts, through soil feeding 11,000 gallons per minute,

were carried down 267 and 216 feet, respectively, in 23 and 20 months, with a cost of $280 and $340 per foot. In the latest application 569 feet were sunk, 16 feet diameter, at an average cost of $143 per foot for both shaft and lining, which latter cost $70 per foot. With a guaranteed success at so low a rate, it is surprising that American engineers, usually so progressive, have not employed this method before acknowledging failures; but no attempt to introduce this plan here is as yet recorded.

The Trepans.—The ordinary procedure comprises first drilling a guiding bore-pit about 50 feet deep and 4.5 feet across; then widening the shaft by a reamer to the required diameter, and alternating these drills with every 50 feet of drilling. These drills, or "trepans," are operated from a walking-beam by a surface engine.

The small trepan (Fig. 196) consists of a blade of forged iron, into the lower side of which are keyed a number of pointed steel teeth, and a stem connecting the blade to the suspension-rods by means of a sliding-box. This last partially corresponds to the "jars" (Fig. 277) of oil-well outfits, takes up the jar, and is an essential element of the tool. The trepans are massive—for hard rock, weighing from 8 tons up—and are raised 6 inches or so, turned slightly for each blow, and dropped; their concussion disintegrates the rock along a diameter of the circle. The progress in flint is 3 inches per day; in chalk, 3 feet; in sandstone, 1 foot, and in coal measures, 16 inches Most of the material is broken quite fine, though 2-inch and 3-inch stuff is not unusual.

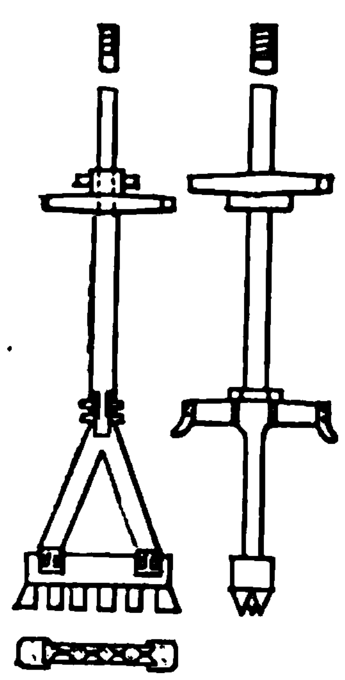
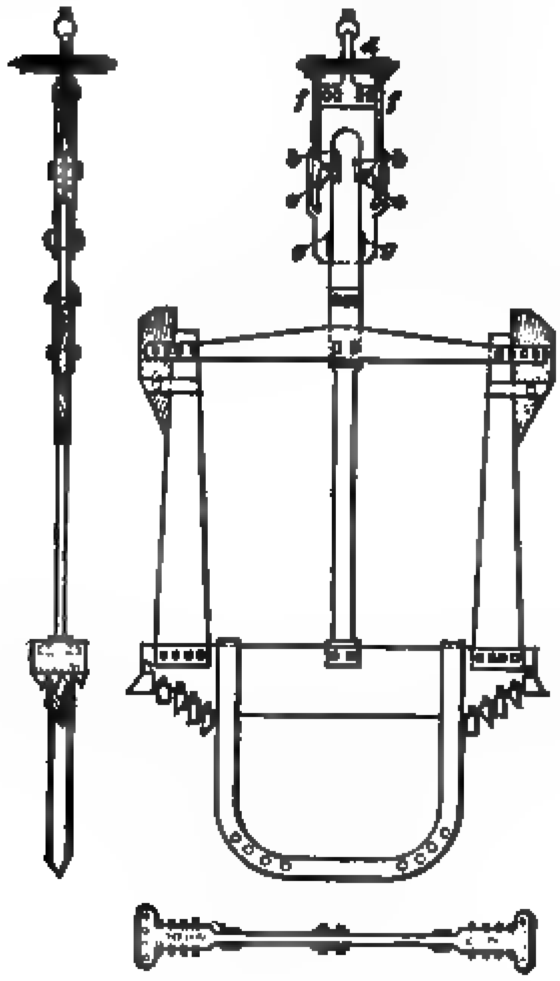


FIG. 196.
A Trepan.

The larger trepan replaces the smaller one after 50 feet of drilling. It is similar (Fig. 197) to the smaller one, but the blade is deeper at the centre than at the ends, so that its teeth cut a base sloping to the centre. The central, toothless portion of the tool has a U-shaped guide that fits the smaller hole. This tool, often weighing as much as 16 tons, cuts the shaft to full width,

or it may be succeeded by another similar reamer, the detritus falling into the smaller hole, from which it is hoisted by the sludger (Fig. 199). In alternate stages the drilling and widening progresses, while the tubbing is subsequently lowered by a separate engine. All these operations being conducted under water, the trepan requires to be automatically kept vertical. Two guides, carrying at the extremities horizontal and vertical cutters, accomplish this marvellously well. The record of the preliminary shaft, 4½ feet in diameter, showed for 508 feet an average progress of 3.3 feet per 24 hours, divided up as follows: 51 per cent of



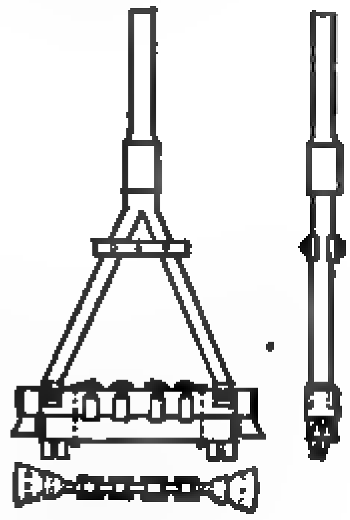


Fig. 198.

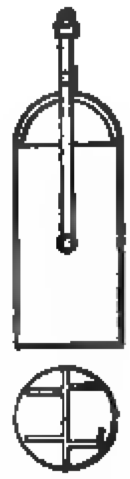


Fig. 197. Fig. 199. Fig. 200.

The Kind-Chaudron Trepans.

the time was occupied in drilling, 19 per cent in raising and lowering the tools, 20 per cent in dredging, and 10 per cent in repairs and delays. Widening the shaft to 14 feet and down 460 feet took ten months; reaming, 46 per cent of the time; altering and operating the tools, etc., 14 per cent; dredging, 22 per cent; delays and repairs, 18 per cent.

The widening trepan reamer and sand-bucket dredge are shown in Figs. 197 and 198.

There are no screws or nuts to loosen; all the parts are keyed to place; but special tools are provided for grappling broken rods, stems, trepans, teeth, etc.

At the surface the operations of boring the pits, building and lowering the tubbing, puddling and sealing the base, are conducted with engines and capstans from a tall derrick, to which extra lengths of rods may be attached with the progress of the drilling. Nine men are employed about the works, only three of whom are skilled laborers. The cost of the installation of the machines, tools, etc., all of which are portable, is about $13,000 to $20,000.

The Tubbing and the False Bottom.—This element of the system is of iron sheeting, built on in 6-foot sections, with leaded joints, and suspended by rods. The flanges are on the inside of the tubbing *bb*, leaving a perfectly smooth exterior, and the joints are true and bolted together. Two sections are lowered daily. As an example, a tubbing 12 feet 7 inches in internal diameter and 280 feet high is quoted; it was 1 inch thick at the top, $1\frac{5}{8}$ inches at the bottom, and weighed 400 tons. The sections were 5 feet high, the flanges $3\frac{1}{2}$ inches wide, 2 inches thick, having leaden wedges between, $4\frac{7}{8}$ inches wide and $\frac{1}{8}$ inch thick, and 20 bolts $1\frac{3}{16}$ inches in diameter.

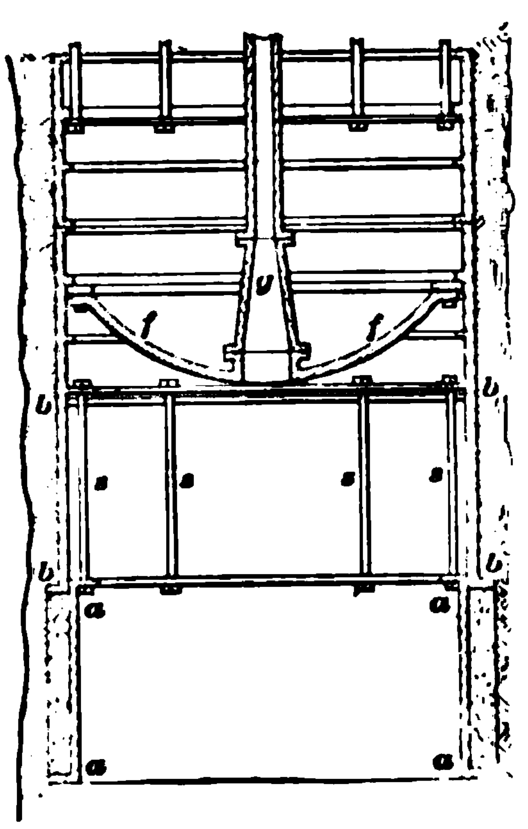


FIG. 201.—The False Bottom.

At the bottom of the iron cylinder are attached two very ingenious appliances, which, operating automatically, have established the process as a success beyond all cavil; the first is a moss-box, *a*, for hermetically sealing the lower end of the tubbing against any influx of water; and the second is the introduction of a false bottom, *f*, by which the sinking of the tubbing is cleverly controlled. These are both adapted to the bottom of the tubbing, as is illustrated in Fig. 201. All the flanges of the tubbing turn inward except the lower one

of the bottom section, *bb*, which is outside, and may act as an annular piston to a lower section, *aa*, of smaller diameter, the upper flange of which turns inward, and the lower one outward. Between these flanges, the moss-box, and the rock, the annular space is filled with moss, which is not, however, under compression so long as the screw-bolts *ss* support it from the tubbing. It operates like the seed-bag of oil-well diggings.

The false bottom *ff* is attached to the tubbing, with the lower sections of which it forms a diving-bell that floats the whole system. The greater the head of water encountered, the more complete the balance, and the greater is the relief to the rods, *dd*, supporting the hundreds of tons of iron. The safety-pipe, *g*, with cocks and plugs operated from above, is an equilibrium column that permits sinking, or rather regulates its speed. Opened at the top, sinking proceeds rapidly, as the compressed air and water find vent; closed, the whole structure is upheld against gravity. When the plugs are opened they discharge into the tubbing, weight it with water, and at the same time release the pressure below. By proper manipulation, perfect control is had over the lowering of the casing.

Making a Water-tight Joint at the Bottom.—When the tubbing has traversed to the required depth through the water-bearing measures, a seat having been scraped for the moss-box, the entire weight is allowed to fall on the annular piston, *b*, by opening *g* at the surface. The moss is compressed to a hard, water-tight mass, the rods *s* gliding in their bearings (Fig. 202). Up to this time the shaft is more or less full of water (the process is independent of the amount of water encountered). This may now be pumped, but usually is not until a cement backing has been inserted and hardened to insure solidity; after that, if the joints of the metallic column are well made, the shaft is perfectly tight, and the mine is insulated from the subterranean current. The introduction of the cement is effected by a closed spoon holding a barrel or so, curved to suit the space.

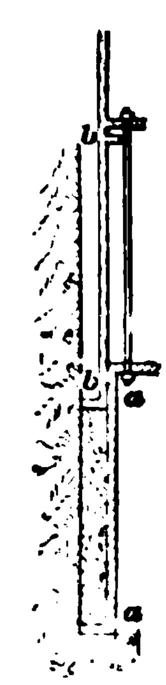


FIG. 202.
The Moss Joint.

Three sets of six men each do this work, burying 400 cu. ft. per day, at a cost of about forty cents per square foot area of lining. A solid foundation of wedged iron curbing is subsequently built on a stout ledge, to take the weight of the cribbing after the other work has been completed.

This method is generally applicable to conditions of soft ground, and especially in watery ground, which can be pierced without recourse to ponderous pumping machinery. Though the pressure of the water is not essential to success, it materially facilitates operations. In a few cases, where the ground was merely wet, not running, tubbing was cheaper than this method. But the facts should not be lost sight of, that none of the delays, perils, and discomfitures of the ordinary methods are here experienced. Its progress is greater, and initially its maintenance is cheaper than other schemes for wet ground, besides never having had a failure, though no shaft of over 14 feet in diameter has yet been sunk by this method.

An objection to the Kind and Chaudron method is that, unless an exact geological section is at hand, there are no means of knowing when the water-bearing stratum has been penetrated.

A short while ago a pair of shafts were sunk in Samlund, Eastern Prussia, for amber, through 147 feet of clay and sand, by a variation of this method. The drill-tools, weighing 1700 lbs., cut a 4-foot 6-inch space, though they had little to do except in the shale-beds. No moss was necessary, as the ground was not wet. Four-feet lengths of tubbing were forced down by jack-screws, each shift with 27 men. The total weight of tubbing in each shaft was 45 tons, and the total cost $17,500.

Lippmann's Drill.—This is a drill of a double V-shape, instead of a straight trepan. It does faster work, as it cuts equally at the periphery with the centre of the circle. With Kind's trepan the blows fell too far apart at the outer edge of the shaft, and too near together at the centre.

Haase's System.—This employs sheeting-piles, in small round iron cylinders driven close together to form a cribbing for the intended shaft. The tubes were about 15 feet long,

¼ inch thick, and 4 inches in diameter, and enclosed an area 10×7, which was then timbered. These were driven through 90 feet of quicksand in five months' time, at a cost of $135 per foot. While standing, they gave good drainage, and did not yield when the excavation of the shaft began.

The Freezing System.—For loose wet alluvium, Mr. C. Poetsch has originated the novel idea of freezing the mass to a solid by boring a great number of holes through the alluvium about 3 feet apart, and lining with copper tubes, inside which are smaller tubes. A concentrated solution of the chlorides of magnesium and calcium circulates through the tubes and freezes the ground, after which the pit can be excavated in the centre of the mass in the ordinary manner, and the tubbing put in. This refrigeration is continued till solid rock is reached. A shaft $7' \times 11'$ was sunk through 26 feet of quicksand, the frozen wall enveloping it being 7 feet thick, at a cost of $190 per foot.

Conducting the Seepage through the Walls.—Water-traps are frequently provided in shafts which are lined with masonry or timber. These consist of grooves cut around the entire periphery of the shaft into a recess in the walling of the shaft and provided with pipes and spouts through which the water can be carried to some more convenient point rather than permit it to give annoyance in the shaft.

If the waters are muddy or contain salts in solution which would clog the spouts, considerable annoyance is given and expensive arrangements must be resorted to.

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CHAPTER III.

TIMBERING ROOMS AND GALLERIES.

The Service of Timbering.—The accident-tables discussed in Chapter XIII show how manifest is the neglect of a few simple rules, endangering life and property; and in no respect is this more painfully impressed than by the mortality record of unpropped rock. Excavations, even in the "rock of ages," cannot be left open any great length of time without support, which, if introduced in time, will prevent disastrous results. Successful superintendents personally watch the timbering and the face-rock diligently, and guard against any springing of the walls. All the effects of pressure are intensified by neglect, and the secret of success is to place timbers before movement begins. Supports are not for bad roofs only; while "awaiting a weak spot, the good roof, so called, catches him," and his stope or room is lost. The eagerness to quickly win the face, while pardonable, promotes avarice, parsimony, want, and then provokes collapse.

Though the conditions underground are such that very simple timbering is required compared with that on the surface, the tendency of the time is toward the employment of special timbermen to make and place the supports. In rooms, driving in soft ground, and the like, the miners must at once prop the excavations. For this reason—and the proscribed space—the character of the timbering should be simple and the sticks light. Fortunately, the pressure of the country rock is inward and toward the openings, and compression, not tension, as on the surface, is to be combated with. This tends to hold the sets together. Tenons and framing may therefore be dispensed with,

except in loose ground, where they are essential for maintaining the integrity of the timbers.

The relative merits of the different varieties of wood need not be discussed here. Oak is undoubtedly the most preferable, but the mines take what can be had in the vicinity. Above "timber-line" we are content with "scrubs." Sawn timber is better than hewn, on account of its greater resistance to decay; and durability is of prime importance to strength. Again, green-wood is heavy; the ordinary 10-inch stick, say 6 feet long, is as much as three men can well handle. Lightness is an essential feature in this most onerous of underground work.

The Life of Timber.—This varies with the condition of the atmosphere and care in dressing. It is rarely as great as that of railroad-ties (twelve years). In many mines head-pieces crumble after two years' standing. Wood rots faster, and shows it less on the surface in dry, vitiated air than in moist air. Alternations of temperature or moisture are very destructive. A cotton-fungus mould is a sure indication of bad air, and, being contagious, requires attention at once.

The decay results from the fermenting of the albuminoids of the sap, the admission of water, and the attack of insects, to which several causes contribute,—bad air, damp air, standing water, and oxidation,—causes all of which are mitigated by an active circulation, and materially remedied by saturation of the pores with some antiseptic. Creosote, Kyanizing, or Burnetizing will give greater life to timber. The timber is placed in a wrought-iron cylinder through end-doors, after closing which the air is exhausted and creosote forced in. Pine absorbs from 10 to 12 lbs. of oil per cubic foot, and the hard woods less. The pressure during the operation is 100 lbs. per square inch.

Timber Consumption.—While it cannot be accurately stated for the average mine, it has been estimated that the timber construction in anthracite mines is 1 cu. ft. per ton product; in the L. S. copper-mines, 1¾; in Leadville and L. S. iron-mines, 3; and in Nevada, 4½.

Every 100 cu. ft. of coal extracted consumes 3.4 cu. ft. of tim-

ber; every ton of excavation in running ground requires about 5 cu. ft. to support the balance. In the Anaconda mines, Mont., 80,000 cu. ft. of timber are used daily; 2000 of such mines would consume the entire forest-area growth. The West Vulcan iron-mines, in L. S., annually consume 2,000,000 feet of lumber and 60,000 pieces of lagging, at a cost of 37 cents per ton of ore mined. In the copper-mines of L. S. this item amounts to from 15 to 31 cents per ton of rock hoisted. So important in the economy of mining and to the safety of lives, the selection and placing of timbers should therefore receive skilled attention; adequate ventilation is equally urgent for their preservation.

Owing to its destructibility and the increasing scarcity of supply that is bringing about the employment of iron and masonry as the certainties of future support. Fortunately, the metal-mines above "timber-line" require but a moderate supply.

Elements of Timbering.—Though the question as to whether the timbering is to be done in a substantial manner at once, or to be considered as provisional, is only answered according to the importance of the gangway; the practice is to assume it permanent. Rooms and stopes are only temporary, and treated as such. Gangways may be subsequently cribbed or masonried; but all pump-rooms, machine-rooms, and stables are very substantially lined.

Each piece should be placed conformably to the principles of the strength of materials, and laid in such direction as will best withstand the pressure whose direction is known. The crushing force is better resisted than is a bending force. They should be placed in the line of the pressure wherever practicable, or in such manner as to act like or take the part of an arch; then, when any movement takes place, its effect will be to tighten the timber in place. Joints should bear the pressure uniformly and their planes be perpendicular to its direction.

Rooms and levels are timbered in 1-, 2-, 3-, or 4-piece sets. The vertical posts, horizontal piece, and top and bottom sill constitute the usual full set, and the inclined stull or cap represents

the one-piece or "quarter" set. Usually the system of room-timbers differs from that in the levels.

Props and Stulls.—Single sticks are used to support the roof back of the men. In long-wall working a large number is used, resting on the floor or on a plate, and hammered into place with a wedge-plate at the roof. They are from 6 to 8 inches diameter, and stand 3 feet apart, in two or three rows, beyond which the roof caves in on the gob. They remain only a few days, are removed by rows to let the roof cave, and are replaced nearer the face (Fig. 3). An average of 70 per cent is recovered; some of the balance cannot be removed; others would endanger the timbermen. Flat caps on top, 20″×10″×2″, are ample

FIG. 203.—A Stull-piece.

for most bad roofs. Slate requires a large plate. It is a poor roof, because it crumbles from the presence of pyrites; sandstone or conglomerate makes a good roof; soapstone is bad; but the most dangerous is fire-clay, which runs when exposed to the moist air. Props come into play (Fig. 210), 12 inches long, 3 inches or 4 inches in diameter, for holding up holed coal. The props do not support the strata above the coal—this the pillars do. They support only a portion of the immediately overlying seams which constitute the roof. The condition of the roof and the method of mining determines the number and distance apart of the props. Every loose block should be removed or propped up.

The presence of seams and cleavages traversing one another in the rock materially affects the selection of the modes of tim-

bering. The parallel joints are not troublesome. Horses in the vein usually require special attention, as do evidences of sigillariæ. The latter occur like truncated cones, base down, and the circular layer in the roof should be propped as soon as observed.

It must be remembered that the prop, as the support for the roof, is employed, not as a means of security, but as a warning of excessive pressure. Its elasticity is of greater importance than its strength. By its bending will be indicated the great pressure

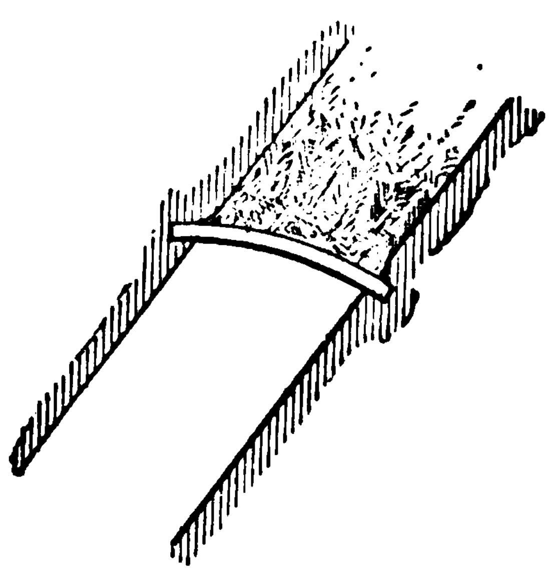


FIG. 204.

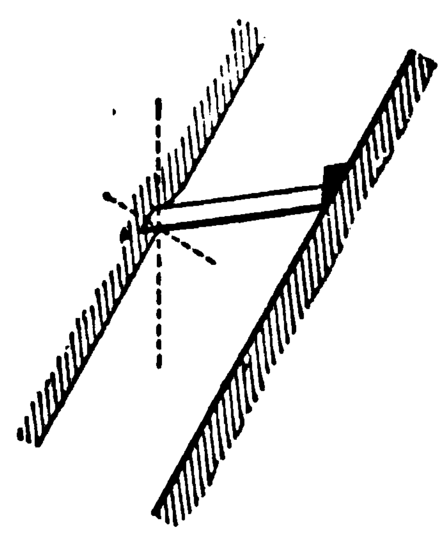


FIG. 205.

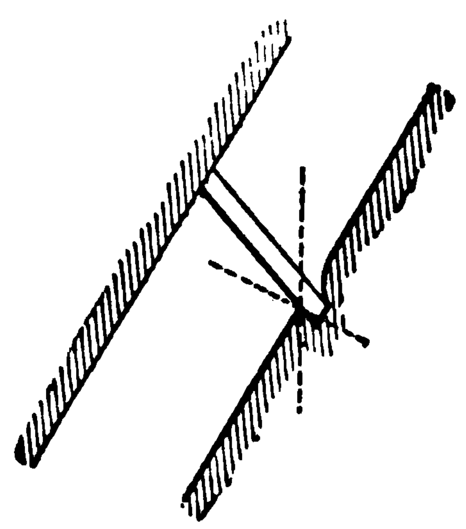


FIG. 206.

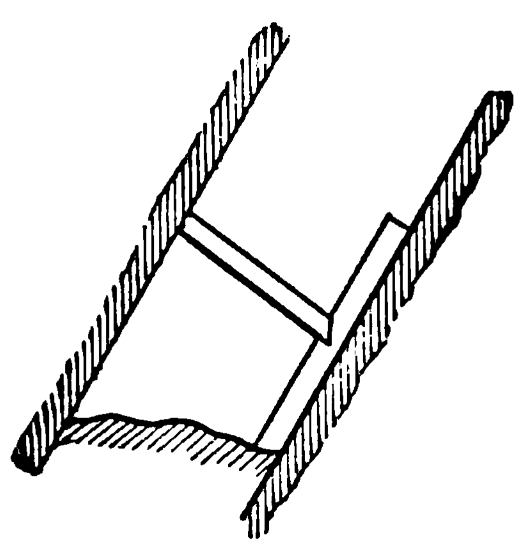


FIG. 207.

Single-stick Timbering in Veins.

to which it is subject, and therefore the necessity for extra precautions, if the room is still to be kept open, or for protecting the miners, if the roof is to be permitted to fall. For this reason props of cast iron or of steel in working places are not recommended, however well they may serve as a secure support for rock in the main travelling-ways.

States vary in their statutory requirements as to the nearness of the props to the face, but 15 feet is the farthest allowed in

any coal region. A distance of 5 or 6 feet is ample space for the men, but not for machine cutters.

In metal-mines the prop is used as a stull (Figs. 203, 205, 206), resting in a notch ("hitch"), generally on the foot-wall, unless the hanging-wall is much softer, and driven into place with a wedge-piece, by mallets. In veins of small inclination the stulls are normal; otherwise they stand between normal and the vertical, because of the combined action of the wall-thrust and gravity. Its angle toward the vertical from the normal is about one fourth the pitch of the vein. They are round or dressed, and of a size and distance apart dependent upon the weight of waste stope-rock to be upheld. It is better to increase

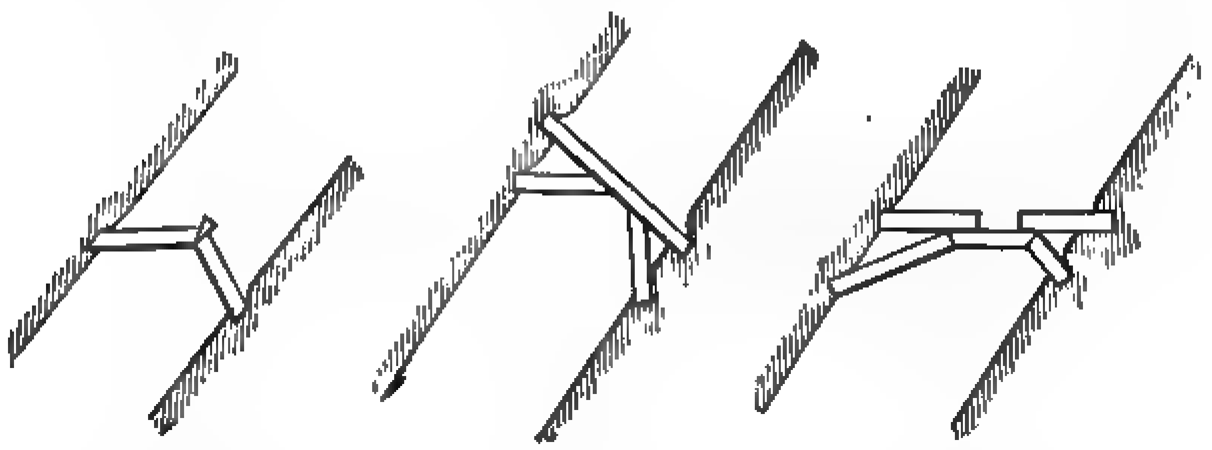


FIG. 208.—Braced Stulls. FIG. 209.—Vein Timbering.

the number than the size, though their strength is directly as the cube of their diameter. $d^3 = 0.05hw^2m$ is the formula for calculating the size of any round stull held at two ends. h is the height of rock along the vein in feet; m the distance between the stulls in feet; w the width of the vein in feet; d the diameter

FIG. 210.—Coal Chocks.

of the stull in inches. To support 60 feet of stull-dirt the timbers are 7′ long, 12″ diameter and 30 inches apart. If reliance is to be placed upon the stull, it must be of ample proportions

and have sufficient bearing to receive the entire pressure uniformly.

Mill-holes.—Steep deposits with good walls will require stulls at distances apart convenient for the men to go to and from their places of work. If there is sufficient waste to furnish convenient footing for the men, no timbering will be required except that used in lining the mill hole and the stulls over the drift to support the waste. Under each mill-hole is a plat, framed for the gate of the chute, with loose boards over the car-track laid upon three horizontal spreaders 7 feet above the track, to afford a loading chute. In wide veins the mill-holes are cribbed and frequently have two compartments, one being timbered for the storage and the other for the passageway.

If either wall is soft, a broad slab or post is laid against it to take the thrust (Fig. 207).

FIG. 211.

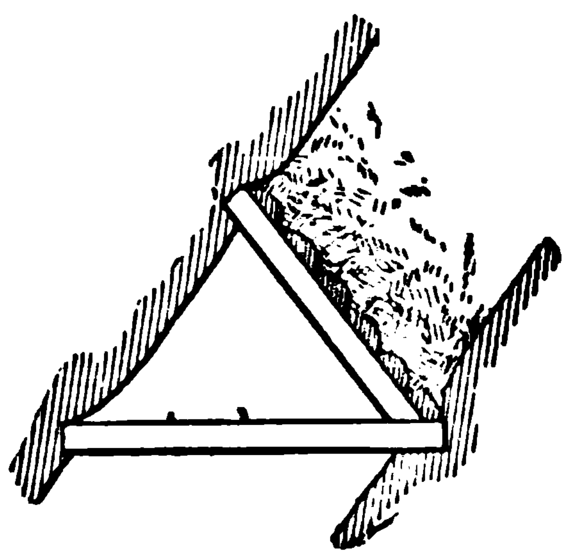
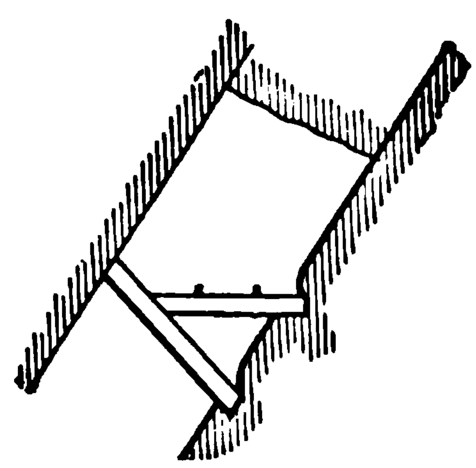


FIG. 212.



Underhand Timbering.

Braced Stulls.—When the distance between the walls is too great for a single convenient-sized stull-piece, an arrangement of shorter sticks, indicated in Fig. 208, is common. A wedge or plank is braced against the walls and extends longitudinally with the drift to be covered. This is not so good as Fig. 209, wherein one or two struts relieve the stull. Figs. 211 and 212 are for flooring in underhand work, and give support also to the stull-dirt overhead. In Fig. 209 the saddleback, or straining-beam, carries the load. Not infrequently the caps may be supported by struts in flat seams, like Fig. 213, or by a single centre-prop in double-track gangways and slopes (Figs. 214

and 215); but, besides taking up room, they are the cause of too many accidents. In fact, much depends upon the cleverness of the men in setting the timbers to the best advantage. For example, a curved stick is beneficially placed if used as shown in Fig. 204. It then becomes an arch. The temptation

Fig. 213.

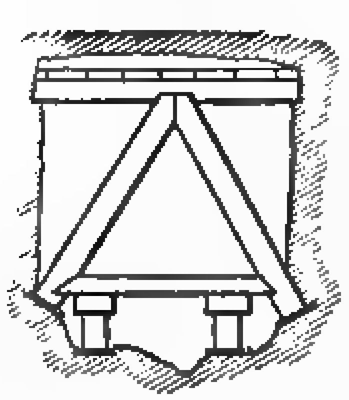


Fig. 214.

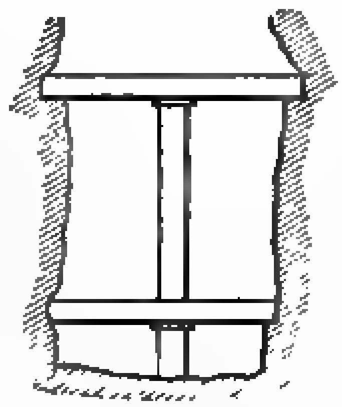


Fig. 215.

The Use of a Centre-prop.

to cut and notch and spike should be restrained, or the continuity of the fibres will be destroyed. The use of wedges should be avoided unless care be taken to distribute the pressure over the entire area of the post or stull.

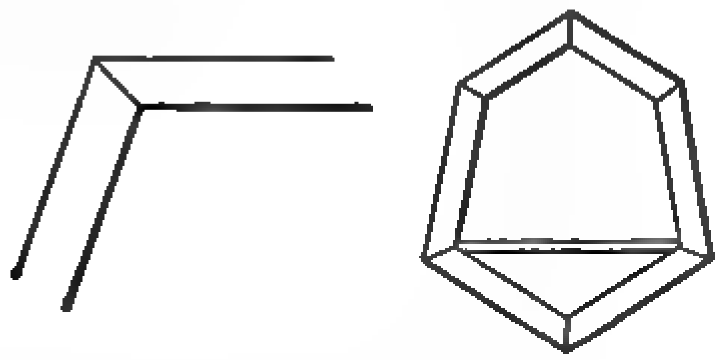


Fig. 216.—Bevelled Joints.

Framed Sets.—The props simply support the roof or the stull-dirt and receive longitudinal pressure only. When the stress is also from the sides, frames are made in sets composed of a cap or collar resting on two posts or legs studded on a sill or sleeper. The trapezoidal form is stronger than the rectangular, and is equally serviceable for carway. This form is susceptible of many varied modifications of shape, frame, and joints.

Joints.—The joints employed in gangway sets where no movement is expected are the flush or butt (Figs. 218 and 219), cut

with precision. Whether the sticks be round or square, the joints should be flat. Never should a round cap be made to rest in the hollow of the post (Fig. 217), for the fit cannot be made perfect nor the splitting of the post averted. The cap should be shouldered to bear flat on the leg (Figs. 218 and 219). When the cap receives vertical pressure only, its entire width bears on the

FIG. 217.

FIG. 218.

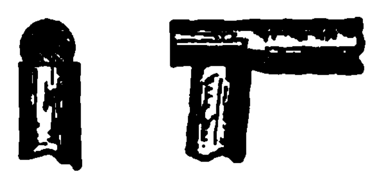
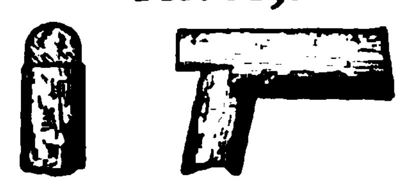


FIG. 219.



Flush-joints.

legs, as in Figs. 218 and 221; if the pressure is partly from the sides, the joint is dressed to form as in Fig. 219; for Fig. 220 the lagging and backing must be firm. The prop and collar-joint (Figs 222 to 225) are simple and effective; the bevel-joint is not uncommon in mining-work; Fig. 267 is an elaboration of it, seen

FIG. 221.

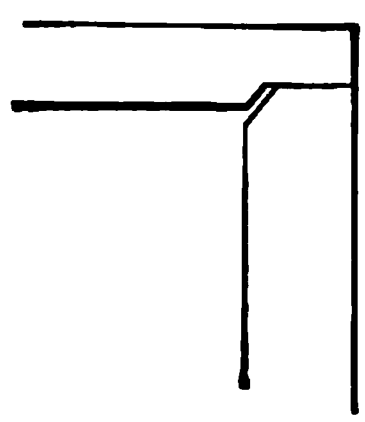


FIG. 222.

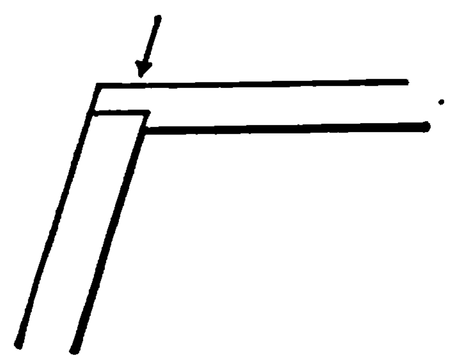
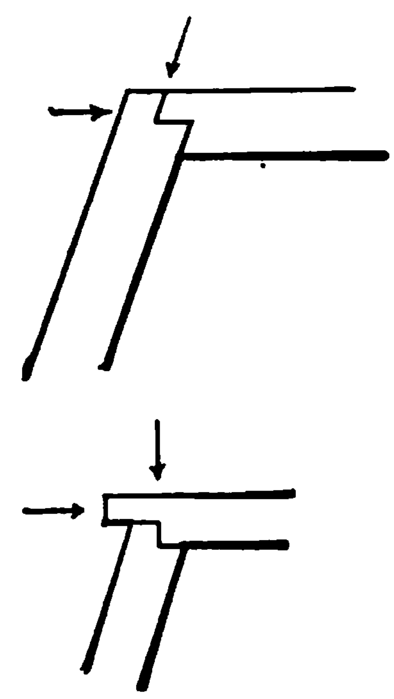


FIG 220.



Shoulder-joints.

in large tunnel-work, but is a very injudicious concentration of pressure at one point, the avoidance of which is the very design of framing. The mortise and tenon joint is rare in underground work, except in pump-rooms and the like. So, also, there is little use for the scarf-joints, unless perhaps in building beams for arch centres. Wedges and head-blocks are essentials in the tightening of frames and to lengthen timbers. Their

removal eases the work of reclaiming the sticks whole. The joints ought to be tarred for effective preservation.

Except as clamps, dogs, staples, bands, and spikes, little iron is used in underground work—perhaps 1 lb. for every 100 cu. ft. of lumber placed. Timbers near to blasting operations are often fastened, and bands used around them to prevent splitting or derangement; otherwise the use of iron is not commended,

FIG. 223. FIG. 224.

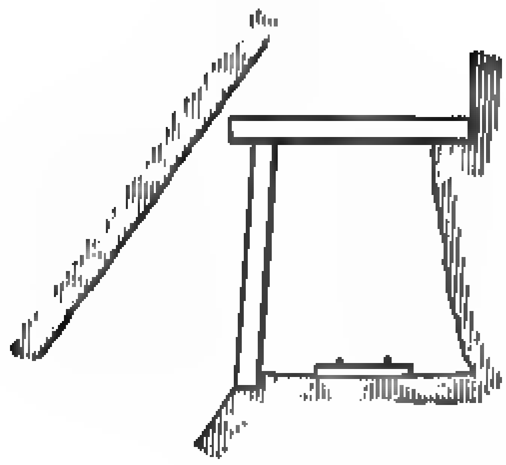


FIG. 225. FIG. 226.

Two-piece Sets.

except as auxiliary fasteners, for centres, etc. Iron rusts rapidly when in contact with ligneous matter.

The Dimensions of Sets.—Sets are of a clear height of 5 feet 6 inches or 6 feet 6 inches, and a width dependent upon the number of compartments. A single way is about 4 feet wide, though this leaves little spare room. The narrowest heading in which miners can work conveniently is 3 feet wide and 5 feet high. A width of 5 feet at the bottom, and at the top of 4 feet, is ample for all purposes of a single way. A double way should be 9

feet wide for men and cars, and a three-compartment way a little wider.

Though the total cost per lineal foot may be somewhat greater for a wider drift than for one smaller, the cost per cubic yard of broken rock is less, and the difference in the cost of timbering is slight, but the gain in rapidity of driving markedly favors large tunnels. Again, as fully 12 per cent of the mine fatalities are from crushing by cars, not only should ample room be provided for passageway for the men, but numerous niches for retreat.

In the standard coal-seams a great height is not possible,

Fig. 227.—Reinforced Two-piece Set for Wide Vein.

though not infrequently the roof is ripped to secure mule height. When the gangway carries a ventilator-box or gutterway, greater height is required, and the timbering for the purpose is illustrated in Figs. 228, 230, and 231. The diameter of he timbers used is as small as will serve the purpose, though it is often 1.4 inches. Large timbers are often needed, but not placed, because inconvenient to handle. Instead of the building of thicker pieces, the sets are placed nearer than the average—3 feet 6 inches; skin to skin is not unusual in shattered ground. The height of slopes depends upon the mode of haulage The use of carriage requires great height; with a dip of less than 40°, the height is about 7 feet. For skips, the normal height of 6 feet will do

Gangway Sets.—Four pieces usually constitute the frame, but the sill may be dispensed with where the floor is good. In slopes sills are essential, wedged into place to secure the trackway. In laying the sills the trench is dug lower in the centre than at the ends, so they will not break. Upon their ends the

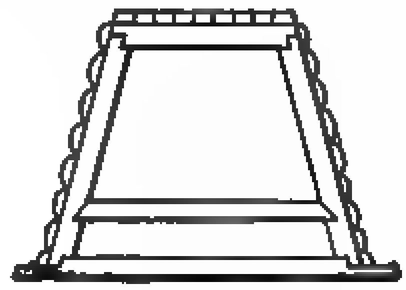


Fig. 228. Fig. 229.

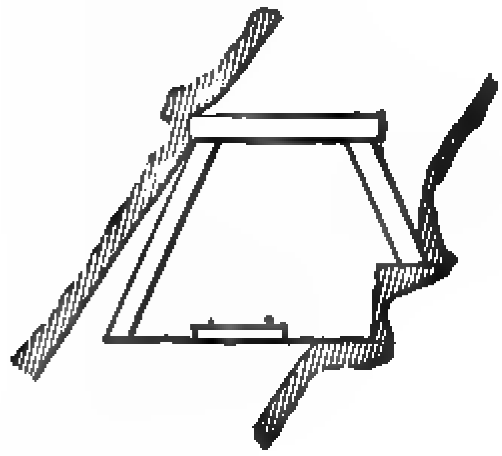


Fig 230. Fig. 231. Fig. 232.

Three-piece and Four-piece Sets.

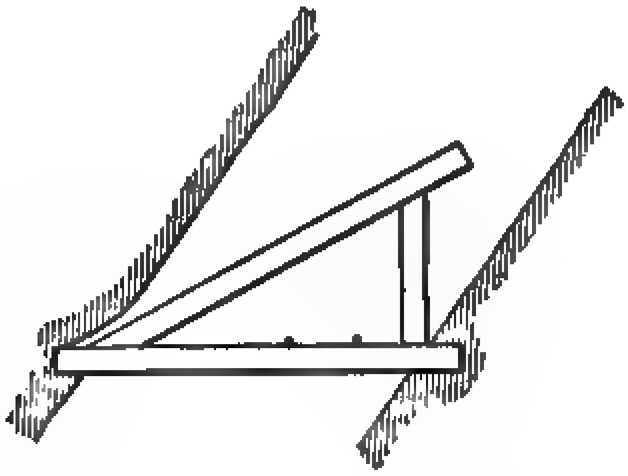
posts rest to support a cap, which is wedged into bearing by blocks and packing.

Where a level road is driven through firm material, only the roof of which needs support, the post and collar form of timbering (Fig. 236 or 223) will suffice. In pitching seams a variety is employed, as in Fig. 232, when the vein and country rock are sound and the hanging-wall soft. With a poor roof and firm vein-matter one leg is floored (Fig. 227), and the other rests on the vein. This is also seen in coal regions. Fig. 233 is a strong form for pressure from sides and top. For such conditions the arched form of short timbers may be utilized as in Fig. 204.

Sets are laid close together, or the distance between them is lagged.

Timbering Loose Ground.—Should the vein-matter be too loose to stand up, the gangway is lagged against a long brace

Fig. 233. Fig. 234.



Wide-vein Timbering.

(Fig. 227). In wide, soft veins a similar idea is employed, as in Fig. 234.

Gob-roads are timbered only at the roof, by the lagging of caps, resting on dry pack-walls. Occasional chocks will give

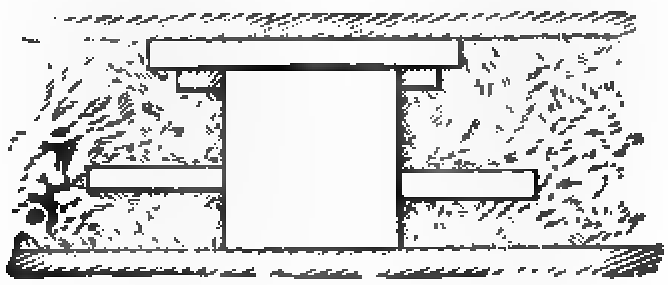


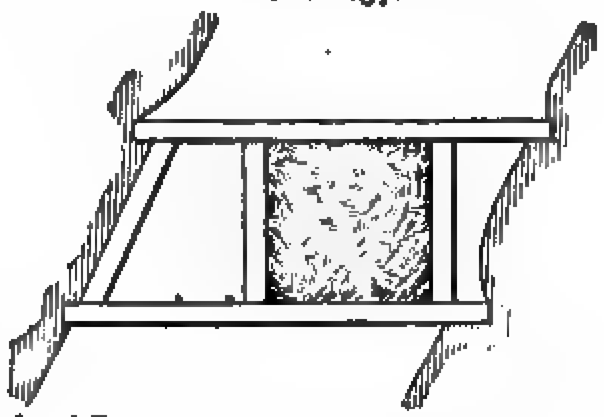
Fig. 235.—Timbering a Gob-road.

greater consistency to the whole (Fig. 235). But as the subsidence of the roof cannot be prevented the road cannot be kept open without repairs until the gob is pressed solid. Meanwhile the maintenance is a serious item. The plan of keeping a road open along the face of coal through waste is never to be commended. The mode of timbering alluvial gold-mines, called "blocking," is said by the inspector of mines at Sandhurst to be perfectly safe. The prop and collar system is used while passing the chain pillar on either side of the gangway; beyond, a modification, in which each cap is, in turn, made to rest upon the caps of the preceding sets, already built, and upon the collar

of the next. The timbers are 8 inches in diameter. In very soft ground each cap has two posts for its support.

Timbering Roads in Large Deposits.—This is difficult. The gangway takes only a small portion of the width of the vein; the rest, if firm, is left to stand, or if loose, packed with waste.

FIG. 236. FIG. 237.



Special Methods of Frames.

In the Great Devon Consols mine the compartments are used one for "attle" (waste), and the other for travelling (Fig. 237). The vein is 22 feet wide; the stulls are 20 inches and 18 inches. In a 24-foot vein a sole-piece of 24-inch timber and struts of 18 inches with a longitudinal piece at the apex, made a very

FIG. 238.

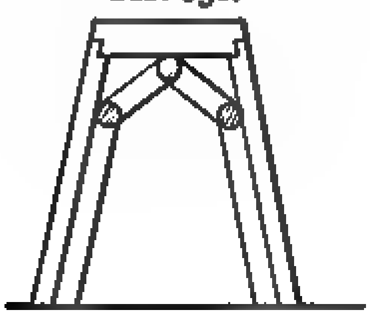
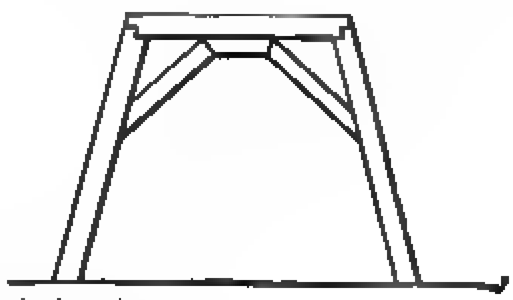


FIG. 239.



Reinforced Timber Sets.

strong frame (Fig. 234). With 3-inch plank lagging, from 10 to 50 fathoms of attle are carried. The hitches are cut 18 inches deep. In Southern France, with great pressure from the roof and for supporting heavy waste, Figs. 238 and 239 are much seen. In many cases an arch of vein-matter 10 feet thick remains untouched from the stope below. With a system of filling this arch is subsequently recovered. In fact, without filling, no large deep mine can be held by timbers.

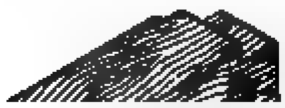


Fig. 240.—Square-set Timbering.

If the vein-matter is very soft, the character of the framing must be entirely altered. In the Austrian salt-mines the problem is very difficult, because the material assumes the nature of a fluid. In rock that decomposes upon exposure to the air the timbering involves some elaborate form of framing. The gangways in a vein or bed may then be a component part of the square-set system (Fig. 240). Where ground has a tendency to swell, the only way to save the timbers is to ease up the ground behind

FIG. 241.—Heavy Timbering for Galleries.

the timbers from time to time until the ground settles to its natural state. The swelling can neither be prevented nor resisted. For shifting ground, the style of Sutro-tunnel timbering (Fig. 241) is preferable to that illustrated in Fig. 268, which has proven insecure.

In relieving timbering and tunnel linings of the result of continual movement an admirable arrangement consists in driving a lateral tunnel or two with heavy timbering, but having an open

face to serve as a safety-valve through which the excessive pressure will be expended.

Few cross-cut tunnels require timbering further than 50 feet or so from the mouth, for the ground stands well. In pitching rock and porphyry the roof should be heavily braced.

Lagging.—If the spaces between the sets cannot be left open, the sides and top are lagged with plank or "slabs" from the

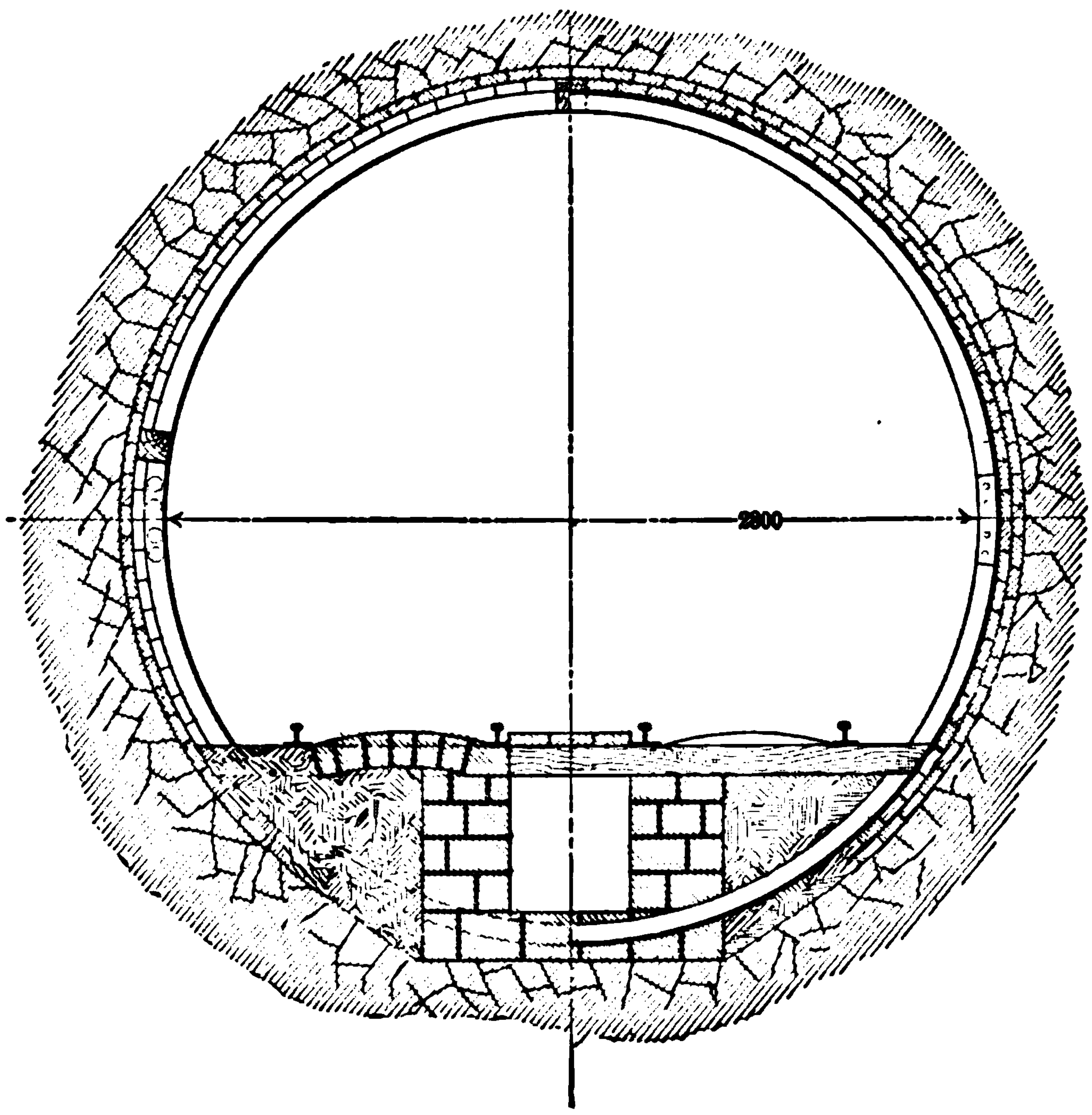


FIG. 242.—Iron Lining for Double-track Gangway.

saw-mills. These are driven close—with the flat side out, though they are better reversed. Lagging should never be very strong; the slabs are always weaker than the members of the frames, and serve merely to prevent the fine rock from sifting into the gallery and leaving an open space that gives opportunity for movement, which if once begun can never be resisted. A very slight movement produces sufficient pressure to break the

lagging, and thus relieve the costlier work. These are readily replaced on occasion. In many cases round poles are used, 3 or 4 inches in diameter, overlapping two sets. If the roof does not run, a few logs suffice (Fig. 227). In soft ground or under poor roof the men are protected, and advance made by "poling" ahead of the face (Fig. 271). The open space between the lagging and the rock is packed with waste or wedged perfectly. Brush piled back of the lagging holds up the "smalls" well.

When the capping over the firm ore of a flat-bed is not good, timbering may be saved by not stripping the entire height of the vein, but leaving a layer of it for roof to be removed later.

The Use of Iron Underground.—Of the relative merits of iron, wood, and mason y much is heard. Suffice it to say, that in Europe, where the utmost care is taken to preserve the timber —by replanting for each tree cut down—iron and masonry are put to extensive service. As to props of metal or of wood, the number would be the same—more for a brittle, less for a flexible roof; and whatever the condition of the roof, the size of the metal props would be nearly the same Iron props are of the + or O cross section, set on the thill or upon a foot-block, to be drawn by lever or bar and chain. Jack-screws have been used, but their expense is too great for their general use. Cast-iron props, auxiliary to pack-walls, 9 feet apart, 5 feet long, 4 inches outside diameter, weighing 150 lbs., have been employed in collieries; their use is emphatically stated to be cheaper than wood.

Levels are not infrequently lined with iron tubbing similar to that adopted for shafts

The increased price of timber for underground work has induced the use of steel. The common structural sizes of steel are used in the galleries. The sets are usually of the three-piece type and the structure is of the strap type of connection. I beams or channels are used for the caps and posts, the latter being bevelled for the batir desired. The connections are made by angle straps, bolted or riveted into place. The foot of the

posts is usually set into a cast shoe upon a cast base. In some mines the pieces are connected by pins instead of angle straps, the latter being cheaper but more difficult in connection. In choosing the size of the members the heaviest shapes are usually taken in order to obtain the thickest possible web and thus a small size of the pins. When channels are used in pairs with gas-pipe separators, they have a special socket for the foot.

Masonry Lining for Roads.—The use of masonry for drifts and tunnels is very common. Where timber is rapidly destroyed, or where the pressure is too great for rectangular frames to be economically employed, the arch is pressed into service. On the other hand, good stone must be plenty and cheap. It is laid dry or cemented, with the walls straight or curved in the top arch. The principle usually followed is to turn the arch out-

FIG. 243.

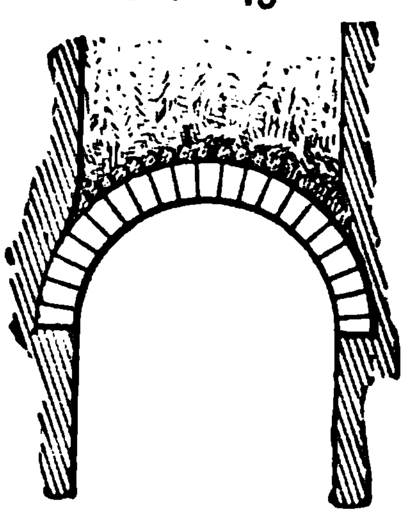


FIG. 244.

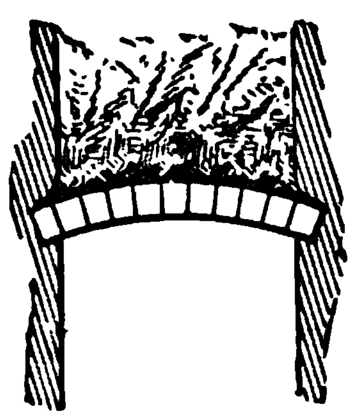
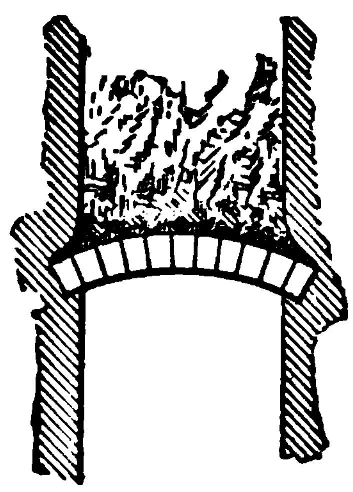


FIG. 245.



The Segmental Arch as a Stull.

ward against the direction of pressure. The top and the floor may be arched and the sides in four separate curves, or they are combined into one geometrical figure, either the egg shape, ellipse, or circle. In some mines the masonry is built exclusively in all the openings, dispensing entirely with the use of timber In others only the main underground arteries are walled. Generally the masonry is built after preliminary framing has served for the exploration work. The walls may be continuous along the entire length of the gallery or may be in sections where the wall-rock is not strong enough to dispense with auxiliary support. The sides of air-crossings require walling, as also the spaces where ventilating-doors are fixed between their posts and the sides of the roads. Whatever the form of the masonry or lining, it is

indispensable that the spaces back of them shall be filled with waste rock and contain no decomposable material.

Where the roof is firm and the sides only are weak, straight walls are built of stone or brick from a foundation a few inches

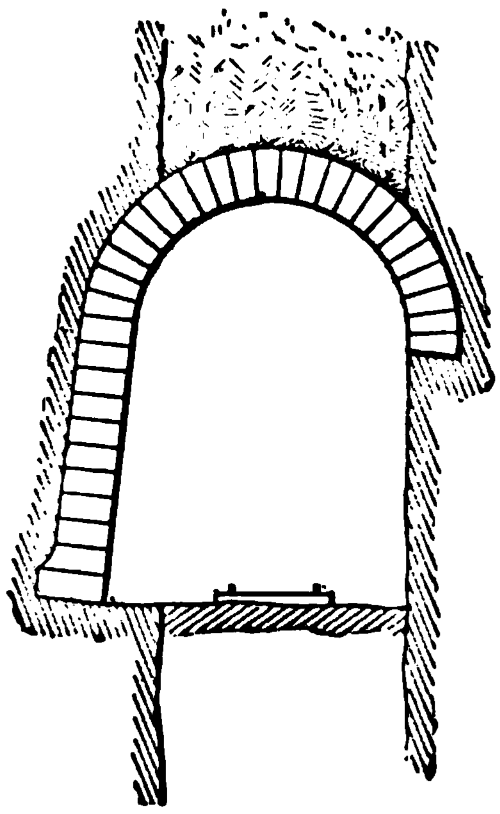


FIG. 246.

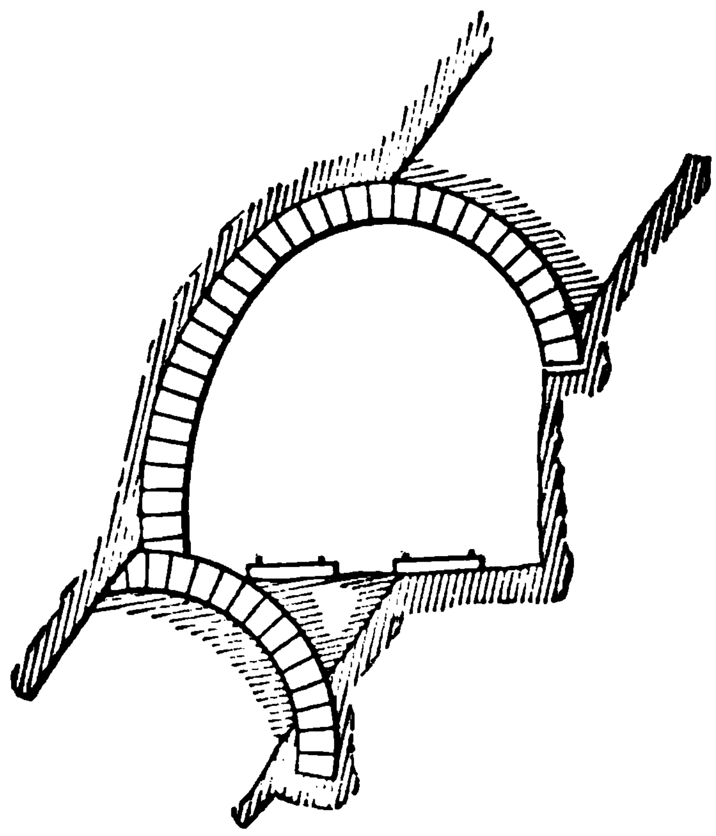


FIG. 247.

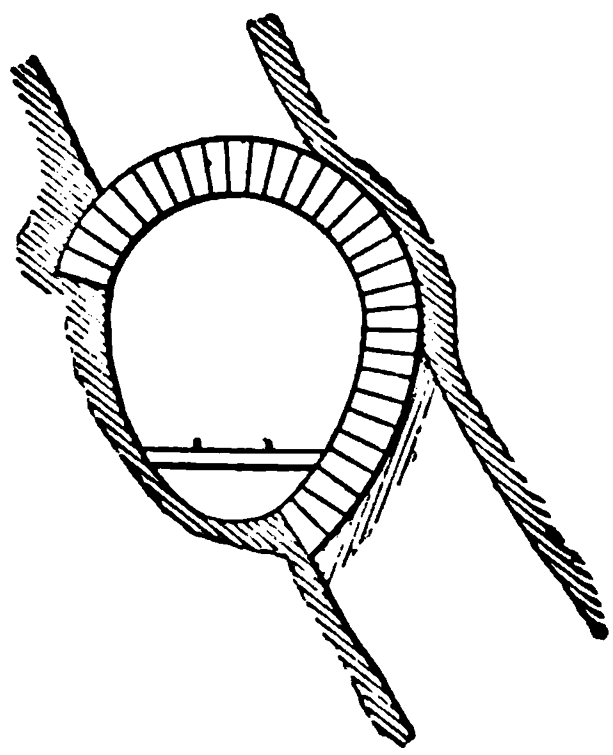


FIG. 248.
Masonry Arches on One Soft Wall.

below the floor line to the roof on either side (Fig. 252). If there is a slight pressure from the sides the walls may be inclined or receive a batir to give a secure base. Upon these walls timber caps or struts covered with lagging are frequently employed to support the roof instead of leaving the roof undisturbed.

The arch is preferred for all permanent ways. It may be segmental or semicircular for the support of waste in stopes, according to the weight to be upheld. They are seldom more than two bricks thick, and when properly laid should be able to withstand all the pressure which emergencies require.

In Figs. 246 to 248 are illustrated arches when the vein-matter and the hanging-wall are too weak to be self-supporting. Firm

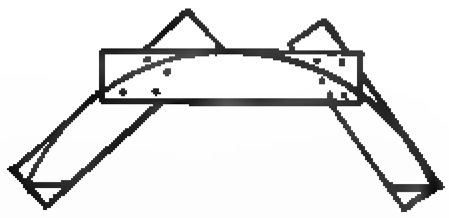
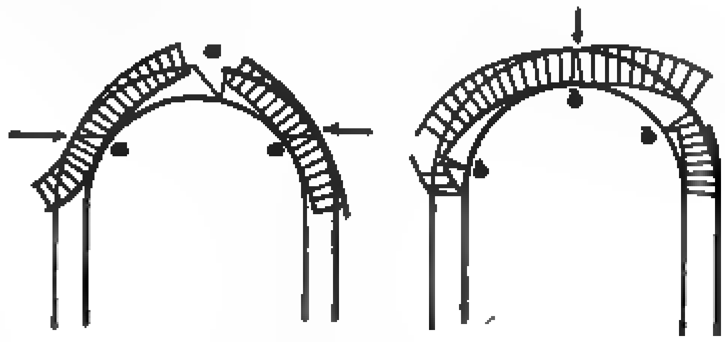


Fig. 249.—Arch Centre.



Figs. 250 and 251.—Arch Failures.

seats for the arch must be provided or imposts are built, as in Fig. 247.

Arch Centres.—Whatever the procedure, arches are built on centres and by template, for invert and walls. The centers should be made of light, small, easily framed sticks, that are not so close as to interfere with work, yet strong enough to sup-

Fig. 252.—Walling Gob-roads.

port the thrust that may fall on them when the tunnel-timbering is removed. Its shape may be whatever is the most convenient for the traffic. The elliptical linear arch is, however, the form most commonly adopted, the side and roof comprising the upper part of the ellipse, which is closed below by a segmental invert arch, with the springing lines on horizontal faces. In stratified rocks, the strongest form for the roof is that of a pointed arch.

Sometimes in solid rock the horseshoe form is used for the top and sides, the floor being level.

The two accompanying figures, 250 and 251, may be interesting as suggesting the places of weakness with the given conditions of pressure. If the excessive pressure is from the top and any opportunity for bulging is given, the collapse will take place as shown in Fig. 251. If the side pressure is very great and the roof resistance small, the break occurs at the keystone.

Tubular Walling of Galleries.—In German mines will be seen examples of tubular walling of the elliptical (Figs. 253 and 254) or inverted oval form, the choice between them for greatest strength being still an unsettled matter. The latter gives greater width at the bottom and smaller area for pressure at the top. Masonry cannot be built unless the ground is previously timbered, or firm enough to stand while the mortar is drying. In soft ground the level is driven by spilling, which can only be replaced by the masonry retreating toward the shaft. When the temporary service of the timber has been accomplished, and masonry is to be substituted, the uprights are cut at the foot, the sills and spilling laths (Fig. 273) removed, the bottom arch is made first, side walls next replace the posts, the cap being temporarily supported on props, the centre set up, and the top laid. Figs. 255 and 256 represent a masonry walling employed where the floor is sound.

FIG. 253.
Masonry Walling.

Dams.—Dams for keeping back water are either straight-backed (Fig. 257) or arched.

The use of iron is advocated and receiving ready acceptance in mining as well as tunnel-work. The life of timber is short, its resistance low, and the component parts of the frame must be rigidly connected. The ordinary constructive forms of iron are applied in the ordinary way for columns, caps, or arches.

Timbering Soft Ground.—In soft ground, which is liable to run, some form of stout framing is indispensable. Indeed, ore-bodies of large, irregular dimensions are peculiarly adapted to

some form of cribbing or square sets. The latter is a natural extension of the system of running contiguous parallel drifts in the ore in two or more stories. In the creviced matter of the Comstock mines, in the very poor ground of the Lake Superior region, in iron-mines where pillars cannot be trusted, in the

FIG. 254. FIG. 255. FIG. 256.

Stone Walling for Gangways.

rotten lead-ores of the Leadville beds, and in the deposits at Butte, Montana, an extensive system of framing is in use and has found ready acceptance in various sections of the world. As a distinctive method of mining, it was referred to in page 47, and though many properties are substituting a filling method, it still retains a hold on the mining fraternity that qualifies it

FIG. 257.—Dams.

for a place here. Still, several causes hasten the decline of its popularity.

The plan is devised for soft ground, and is equally well adapted to large openings in rock with a small tendency to cave, slide, or swell. The system is a simple and natural extension and refinement of the practice of running contiguous parallel drifts in the ore and in two or more stories when the thickness of the ore required it.

The Square Set.—The square set in stoping requires vast quantities of timber, and its framing is also expensive. Nevertheless, its adaptability to all forms of excavation commends it. The timbering of shaft, slope, and slopes may all be rigidly connected in one system with facility. Though there are objections to it, nevertheless it frequently proves in the end to be cheaper and more economical. The open cells of timber may be inserted in the event of an unexpected thrust.

The set is made up of posts, cap, girt, and frequently a brace, which may be the middle number of a set of three pieces forming the letter *N* inside of the main set. The cap usually is placed across the deposit, while the girt, also resting on the post, extends longitudinally with the roof of the ore-body. They are usually framed for 7 feet of clear height to allow of reinforcement sets being placed later, and still to leave ample passageway. The posts are 5 feet apart in the clear.

The usual plan consists in driving the heading along the level and near the centre of the deposit, from which is extended the timbering. Two sill-pieces and girts are placed, then the posts; after which a cap across the drift and girts connects this set with the one last placed in position. The frame is blocked, plumbed, and wedged against the back of the drift. As fast as excavation proceeds on either side of the heading, one-post sills are laid on the floor and penned to the two-post sills, a stull-girt, then a post fitted on that, the cap and girt holding its top in position. As soon as wedged against the top and the side, two-inch plank roofing is laid from cap to cap on top as a floor for the upper set.

This continues to the foot-wall, along which the sets are carried by means of a cap-sill, which combines the functions of the two pieces as shown in the illustration. From this a new line of sets is carried up indefinitely.

Thus each one of the full square sets encloses a rectangular volume or cell. Each post supports two cross-sills and two caps and rests upon four sills (Fig. 258). Note also in Chapter III, Part I, the method of square-set timbering.

The frames are built up as fast as the work is opened, unless it happen that the ground will stand a while, until the timbermen can attend to the chamber. In ore that will not remain in place long enough to advance a set, an intermediate false set of cap and uprights supports lagging overhead until the men reach the full length of a cut. If the ground will not allow of this advance, it runs, and only caving or filling is admissible.

FIG. 258.—Square-set Timbering in Rooms.

The method of procedure varies in different regions, or perhaps with the nature of the ore. The lower floor is worked out first, the bents being added on at the right, left, and ahead; after which the next tier is set in the same manner directly over the first. A species of overhand stoping is sometimes employed whereby the floor tier is progressed only a set or two before the next upper is advanced, to be followed later by another set above, and so on up. The order is a matter of indifference provided a

perfect alignment is secured. Where this method is used in steep veins, very careful surveying is necessary to carry up the tiers of the lower level to a line with those in the upper stope.

When a sill-floor is to be laid upon a large body of ore to be mined from the next lower level, the sills should cover several sets to render less dangerous the work of connecting the lower nest of timber with the upper. The plates against the walls should also cover several sets, if the walls are bad or crumbly.

Sticks 10 inches in diameter or square are used for 7-foot sets, and some as large as 14 inches are not uncommon with 9-foot posts. They are dressed to a faultless fit, being cut to template by some saw like the Hendey (Fig. 260).

When the permanent roof is reached, lagging is laid, wedged, and packed.

Reinforcing Square Sets.—Where the posts cannot be placed in the direction of the greatest pressure, reinforcing sets are used in heavy ground, with the planes of the inclined piece in the direction of the greatest pressure. The sets then assume the form of X or N.

Objections to the System.—All the sets are dependent upon one another, and therefore an absolutely perfect fit is essential to transmit the pressures equally and to maintain the framing intact. If the joints are not true, each one will have a slight play and an opportunity for movement will be afforded. The caps may slip off the posts. The sets should be "plumbed" occasionally to watch for incipient displacement. Sometimes, to prevent this form of failure, solid cribs of timber, two sets wide, are built across the ore-body from one wall to the other.

The sets may crush when the safety-valves of lagging have been omitted. These consist of frames or lagging, slightly weaker than the timbering, arranged to be easily replaced. Felsite or trachyte tends to swell and crush timbers, and in such ground should be closely watched. At the first sign of fracture of any member of the system it is advisable to at once withdraw the men from that locality; for ruin spreads very rapidly after the destruction of a single member of a framing so loosely con-

FIG. 259.—Framing- and Tenoning-machine.

nected as is the square set. A chamber 90 feet high, with 13 tiers of timbers, was in complete ruin within thirty minutes after the first sign of break of a sill.

Cribs.—Rooms and abandoned large stopes are supported by massive columns of heavy timbers carried to the roof and filled with waste. These "cribs" are also built as an abutment in a chamber having the wall as a back. A good foundation is prepared, of the proper size and shape; two logs are laid parallel, and upon them, in the notches at the ends and at

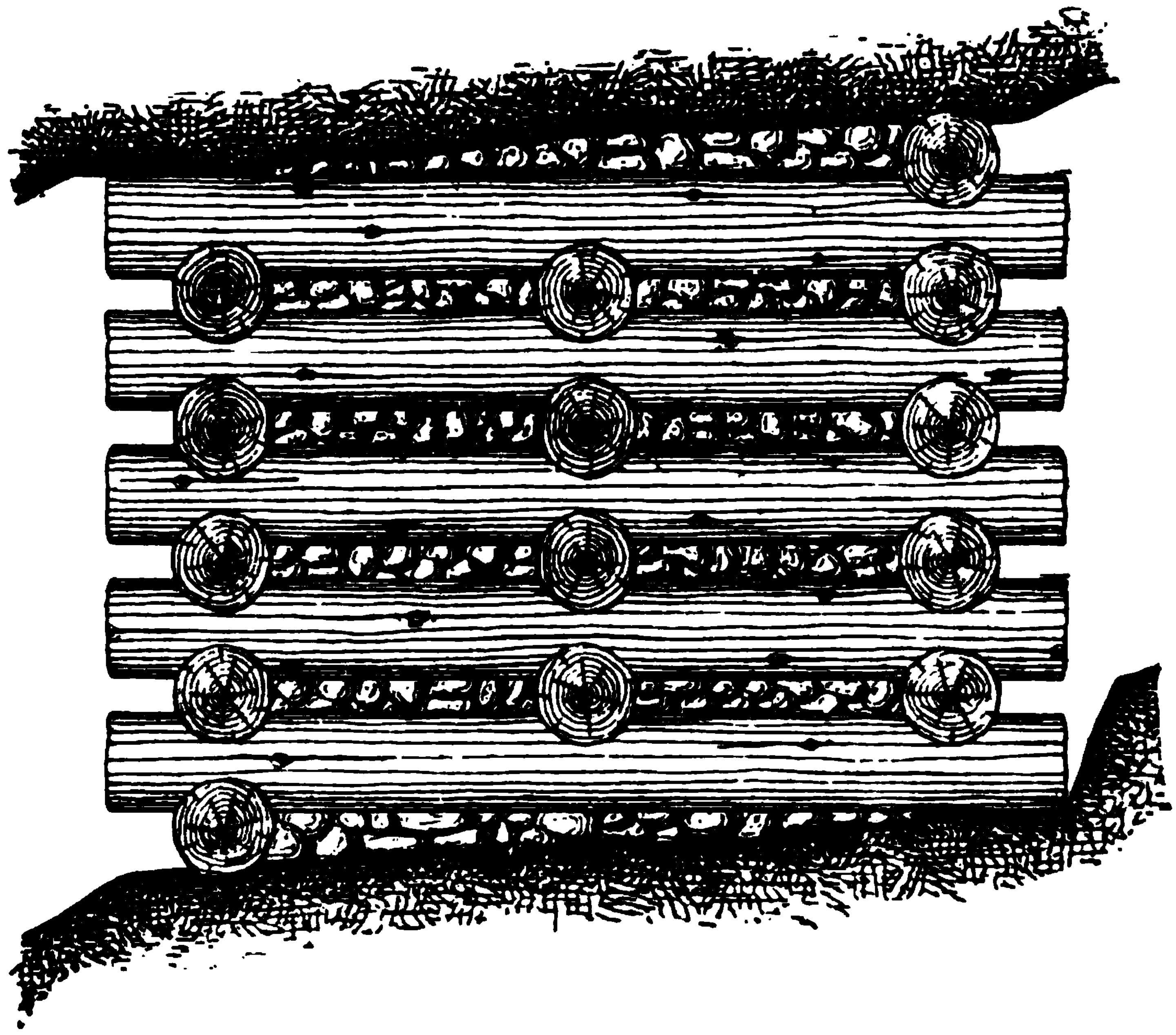


FIG. 260.—A Crib.

the middle, three cross-sills are laid; upon them again rest a pair of sills slightly inside and above the others; upon these, in turn, another layer, etc. (Fig. 260). Inside of this space, as fast as enclosed, waste is piled. The logs are 10 feet and 14 feet long, and 12 inches to 20 inches in diameter at the butt. Cribbing may also be built with the removal of the ore in overhand stoping, the waste being utilized to fill the crib. As the height of

these cribs increases, the area of their base is proportionately increased.

These pens, or cribs, are also used for construction of dams or abutments at mine-dumps, the front face consisting of long face-sills parallel with face of the hill, with cross-sills projecting from the hillside and resting on the face-sills. They may, or, may not, be notched. The structure continues as high as desired the framing being carried up as fast as the waste rock accumulates upon those already laid in position.

Mill-holes are built of cribs 30 inches square in one or two compartments.

It is questionable if there is any choice between the square set and cribbing in large rooms. The former is well suited to turning off into small cells at the ends of the room, but it is dangerous if side pressure exists. Cribbing will never do in very soft ore under brittle roof. This arrangement constitutes a very strong "made" dump, where the rock is not permitted to roll away freely, as, for instance, on account of contiguous buildings down the hill.

Protecting Underground Chambers.—Underground chambers intended for steam-pumps, hoisters, stable, etc., are built in any suitable shape that provides sufficient room, and, being large, require great skill to utilize framing or walling materials to the best advantage. Undoubtedly an arched roof will give the greatest resistance and strength, and masonry is therefore suggested. Besides, the hot, damp atmosphere of steam would rapidly destroy timber. Still, the latter is more convenient than masonry, in heavy sets of three-piece or five-piece arches, and spliced. They are laid close together, lagged over, and packed to prevent wedging apart. The ventilation of these rooms should receive special attention. For air-compressors, engines, and coal-cutters an enlarged level will do, with some stouter caps, railroad rails, and I beams on the wall-posts.

Timbering Landings.—The timbering plats and landings must vary with the character of the intersections. The frames must support one another as well as the country rock, and should

afford firm fastenings for the plats and doors used for the landing of the cars and buckets. The level or gallery should be widened near the shaft, to give room for sidings, storage closet for powder and steel.

The masonry lining of the shaft is supported by an arch to, or by lintel on, two posts or walls at the sides of the gallery landing. The former gives a high opening for landing the buckets.

Mill-holes carried up with the waste are solid cribs of 30 inches square for a manhole, and may also have a compartment for sliding ore (Fig. 87). Either cordwood or sawed blocks are used for the lining. The latter plan gives them greater durability, for the abrasion is along their cut faces.

The timbering of slopes is similar to that of galleries, except that greater care is taken in cutting.

Timber-cutting Tools.—The timberman's tools are few. They need be only hatchet, hammer, and wedge, with a bar and chain for casual work. The timber should be delivered below ready for insertion. A sawmill at the surface is now as much a component of the surface improvements as is the boiler.

Mine-timber Framing-machine.—In Fig. 29 is illustrated a machine which is designed to saw tenons upon timbers of standard size. It consists of a number of adjustable saws which may be arranged at any distance apart to produce the desired dimensions of the head on the timbers. By a machine set to a given pattern, all timbers are finished to an exact shape, which when placed in position ensures an alignment of the framing, the joints being accurate and the contact perfect. Security is thus attained and the life of the frame is prolonged.

Though the work of timbermen cannot accurately be stated, the following is given as the experience of an "old miner" for one day's labor:

Making about 100 feet of wooden ladders;
Placing 65 sq. ft. of close shaft-lining down to a depth of 70 ft.;
" 40 " " " " " " " " " " 200 "
Cutting and dressing mine frames, 50 running feet;
Framing mine-sets, 40 " "

Dressing sills, 30 running feet;
Making 60 poling planks;
" 100 " wedges;
Setting up 20 pairs of props and head-blocks.

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CHAPTER IV.

DRIVING DRIFTS, TUNNELS, AND GANGWAYS.

Levels.—These horizontal openings in metal-mines receive an amount of care commensurate with their importance and service. Like the levels, which also are driven through hard rock, they are classed as "dead work." The difficulties of driving are not so great as for shafts, as their area is smaller and seepage water gives less annoyance. Their location is fixed by conditions discussed in Chapter I. The drifts and cross-cuts of a given mine are of the same dimensions. They serve for one track, and only at the landings are wider to accommodate a second track. They are rarely timbered, for the country rock will stand without support for the entire period of their utility. If they require protection at all, a masonry or iron lining is used.

Adits and Levels in the lode are secured as shown in the previous chapter. Usually the stull and lagging suffice for the average length of time the drift is kept open. Gangways and galleries in coal-mines have a larger area than levels, and being exposed to more treacherous conditions, possess better examples of the timberman's art. Tunnels for railway or drainage purposes must be walled.

Before locating a tunnel of any importance a careful study of the ground is requisite. All the data obtainable from geological reports, borings, etc., should be availed of. The character of the strata, their pitch, and the direction of the subterranean drainage should be known before the character of timbering and the dimensions and amount of timber required can be determined.

In Drinker's "Tunnelling" will be found a discussion of the geological conditions affecting tunnel locations.

The Alignment of a Tunnel.—The axis of a tunnel is kept in a vertical plane by the use of three plumb-bobs (Fig. 261), and its horizontality is tested by a long spirit-level. A grade of two feet in a hundred is given to the floor for purposes of drainage. In collieries the grade is involved with the proposed system of mining.

The alignment of a gentle incline is similarly conducted, though four adjustments are necessary. A saw-cut is made across each sill and 8 inches from the end. Two hubs are set in the sills on this line by transit within 50 feet of the face of the drift or incline. They are lined by a string which must pass through the saw-cut notches. The grade stick is now placed in position to depress or raise the sills to the required inclination, after which they must be laid level and across the drift at right angles with the line.

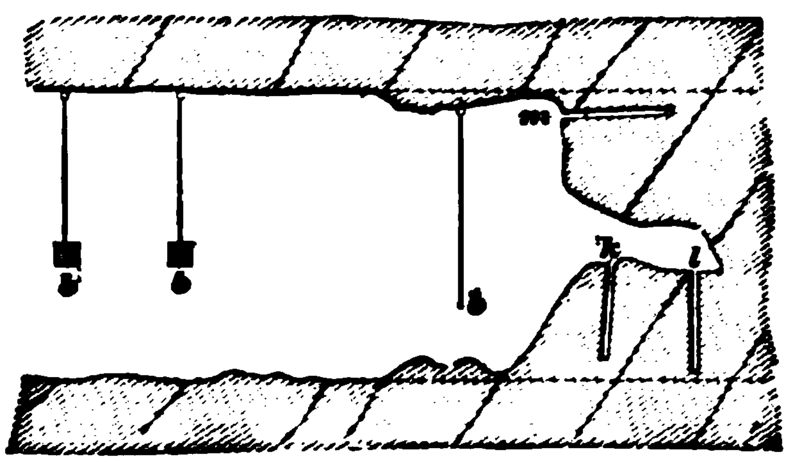


FIG. 261.—Aligning a Tunnel Drift.

Rock Characteristics.—Hard rock presents no serious difficulties to the miner beyond the time and cost of drilling, for it usually affords a good roof and if uncreviced is dry. Granite dolomite and gneiss are of this character. Slates and shales are bad and require arching or block timbering. Porphyry is treacherous, because, though hard when first opened into, it soon decrepitates on exposure. Some of the limestones and sandstones are porous and wet. Clay seams are bad primarily, besides being watercourses for the upper porous strata.

Driving in Creviced Rock.—In uncreviced rock the order of the breaking is immaterial, so long as a good bench is had to shoot from and a favorable working-face is obtained for the

next shot. The crevices and planesi n stratified ground materially assist the shooting. If the seams pitch toward the face (Fig. 262), the upper holes are fired first, or in high galleries the central holes will make a good bench to shoot to from top to bottom. If the cleavage is not marked, or away from the face (Fig. 263), the bottom holes are fired before the upper ones. If electric firing is adopted, the sequence of shots is of little importance. Three men form a gang in driving, and can manage the ordinary-sized drift. In the gangway of 7×7 two machines or two pairs of cutters have ample room. Their progress cannot

FIG. 262 FIG. 263.

Placing Holes in Creviced Rock.

generally be stated. In hard rock a foot a shift is fair, while soft rock can be penetrated at the rate of three feet a day. In shaly ground greater progress might be made did it not require good timbering. In soft ground the advance depends upon the skill of the timberman.

Driving by Air-drills.—It is a matter of indifference where the holes are placed for explosives if machine-drills and simultaneous firing are employed. A drift of ordinary size will easily accommodate one drill, while two can advantageously work in a heading 10 feet wide, obtain more angling holes, and advance more rapidly. In railroad tunnels four are simultaneously drilling. Hand-work is much more depended upon for the driving of gangways, but machine-drills are becoming popular for tunnels. Undoubtedly the progress by machine is greater than by hand, while the cost is nearly the same per lineal foot. Where the drills and air-compressors can be put to use after the mine is opened,

it certainly would be advantageous to employ them in the preliminary and development work.

The consumption of steel in medium ground is about 25 cents per cubic yard removed, though pink quartz will dull 150 bits to the hole, and the blacksmith will consume more steel than the rock. The consumption of powder is about $1 per cubic yard of medium-tough rock removed. This amount will vary with the area of the face and whether the breaking is done by simultaneous or single shots.

Driving Slopes.—The operations of sinking and the timbering of a slope are similar to those of level driving, except that the sill is indispensable to the set. It must be well bedded and let into the rocks on each side to prevent the roadway from slipping down hill. Ofttimes it is stayed by plugs driven into the floor. The sets are also braced against one another by longitudinal studs between the posts at their head.

Driving Tunnels.—Levels and drifts are worked over their entire face. But when tunnels are to be driven of a height greater than 8 feet, they must be broken in benches. Railroad tunnels are of this description, and in hard rock may be driven without any temporary timbering, the benches being attacked like stopes (Fig. 27), where the numbers express the sequence of openings. Frequently, in driving long tunnels, the drift No. 1 is pushed as fast as possible in order to make connection with a shaft or a similar drift approaching from the opposite direction. This communication is for ventilation and haulage purposes.

Numerous tunnels have been driven more than five miles for mining purposes. The Freiberg is 24 miles long; at Clausthal is one nearly 11; the Joseph II. is 9½; the Ernest August, 6½ miles; and the Sutro, 5. These have, moreover, lateral branches that enable them to subserve a great territory. The Gwennap adit, in Cornwall, is said, with its branches, to attain a length of 30 miles. The greatest depth below the surface is not over 900 feet for any one of these.

The Use of Supplementary Shafts.—Auxiliary shafts are sunk at convenient points along the line of a long tunnel to its level;

and from them the excavations are begun contemporaneously with those at the mouths of the tunnel. Sometimes the shafts are to one side of the tunnel line, to keep them free and clear of the tunnel-work. The amount of time allowed for the completion of the tunnel determines the number of points of attack and the number of shafts to be sunk. The latter is also dependent upon the cost of sinking and the hoisting through them relative to that of the long haul in the tunnel. These relations can be mathematically expressed when the several aspects of the case are given. From Foster's Callon's "Lectures on Mining" the following formula is taken:

$$Q = xq + (PS + P'S')\frac{l^2}{4x};$$

wherein q = the total known cost of shaft, x = number of shafts to be opened over a length l, S = the area excavated, S' = the section of the lining, D = the distance between the two adjoining shafts, P and P' = the cost of haulage per lineal yard for each cubic yard of rubbish and of walling material.

$$D^2(PS + P'S') = 4q.$$

The St. Gothard Tunnel, 48,840 feet long, was run from the two ends only; so the Mont Cenis Tunnel, 39,840; the Hoosac and the Sutro tunnels with the assistance of two shafts. The Washington Tunnel, 20,715 feet, had four working shafts. The Rothschenberger mining adit had 18 points of attack along its 24 miles of length. Rziha estimates that the "additional cost of running headings from a shaft is from 5 to 10 per cent higher than running from portals." He also gives a unique table to show that the rate of progress per month is apt to be greater in long than in short tunnels. Of those less than 100 m. long, 29 feet advance was the average of 3; 13 between 400 and 600 m. showed average of 57; 8, up to 1000 m., 114 feet, while 6 between 3000 and 4000 m. had an average of 219 feet, and 4 over 4000 m. progressed 259 feet per month.

Provision for traffic is not a simple matter when it is recalled that, besides the actual mining of an area (say, for example, that of the St. Gothard Tunnel), 26×20, at a rate of 18 feet per day, and the placing of 1000 cu. ft. of timber and 300 tons of masonry per day, 750 tons of broken rock, 15 tons of lumber, and 300 tons of masonry must be handled, loaded, transported, and unloaded at the same time.

As the hoisting through shafts is about half as fast as haulage through a heading, the use of auxiliary shafts is not so economical as it was before the present development of machine-drilling and blasting methods.

Driving a Tunnel.—There are several methods of opening a tunnel, one of which is depicted in Fig. 264. The two side

FIG. 264.—The Order of Attack on a Tunnel-face.

drifts at the bottom may precede the main excavation; in these are built the walls for the roof arch which ultimately lines the tunnel. While the small drift, 1, is being driven to the connection, the sides 2, 2 are following, and in firm ground they may precede 5, 5 at the bottom by 150 feet. At this point the tunnel is being lined with timber or masonry, while gangs are also breaking ground on benches 3 and 4.

If the roof needs support, the plan of work contemplates one similar to that of Fig. 265, with the exception that the central

core 3, 4 remains as a support to the timbers arranged as shown below. Inside of this frame the masonry may be completed, after which the inverted arch, if required.

In soft ground, and especially in inclined strata, the order of driving the benches and the amount and character of timbering varies according to the dip. Circumstances and difficulties are so diversified that no uniform infallible rule has been established for the guidance of the engineer—probably because no system has a superiority over all others for any and all conditions. Safety is an element as important as time and money, and often is a determinant.

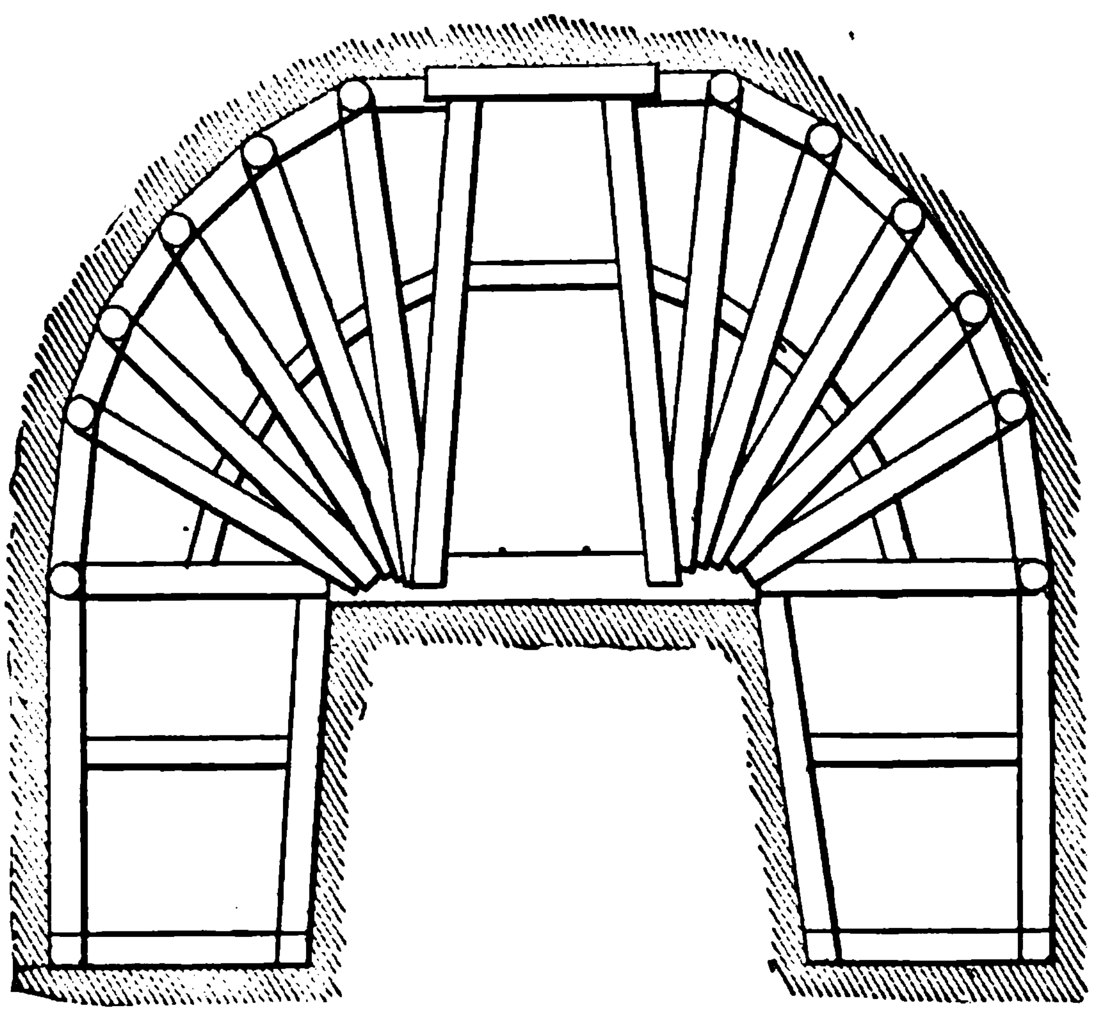


FIG. 265.

Tunnelling Systems.—A brief review of the English, Belgian, German, and Austrian is taken from Drinker's "Tunnelling."

The English system is developed from the experiences in the Thames River tunnel and consists in taking out the full area at once, after the preliminary top-heading has been made, and in supporting the roof by longitudinal top bars while removing the lower section. This gives a full clear area for putting in the masonry, after which, in material having any tendency to

swell, the space behind the side walls is securely grouted. Though built quicker and applicable in 90 per cent of cases, it is unsatisfactory in very bad ground; in heavy ground it requires more timber than does the Austrian, its strongest competitor.

The Belgian method, introduced after the iron shield had been tried ineffectually in quicksand, builds the tunnel as an open cut down to the springing-line of the arch. The arch is then laid, recovered, and underpinned, the bottom removed in benches, working downwards, and the abutments built. Sometimes the centre-core is left, though that is only done in the French and German modifications. The entire area is not attacked at once, but divided into several benches, each being worked separately. The underpinning of the arch may be safe enough in hard ground, but it certainly affords a doubtful security in loose material.

The German system gave rise to what may be called the centre-core system. The work of excavation begins at the foot of each abutment of the arch, where a small heading (Fig. 265) with timber sets is first driven. In this the foundation is laid; above it a second heading, large enough to build another height of wall; above this another, in which the masonry is carried up. The top is then excavated across, and a connection effected between the two sides. In this the arch is completed without the use of centres, while the roof is being supported by stulls or props. The core is then removed.

In hard ground it gives cheap working, from the fact that the core has several faces of attack. In soft ground it is safer, because of the small exposure of roof and face; and the centre-core saves timber. Its ventilation is bad, and the cost of laying masonry is larger than where the masons have elbow-room; it is hard to securely timber, and several prominent engineers have decided against it. Certainly, in soft, treacherous ground, like shales and clays, its defects disqualify it.

The Austrian system admits of mining the whole area in small sections. First a bottom heading is driven and afterwards connected to a top heading, which is finally widened to full width

for a bar-timbering which is carried down to the springing-line. Sides are excavated for the walls, after which foundations are prepared and the side walls built; finally, the invert arch. The cross-rafter timbering (Figs. 267 and 296) used for support admits of the transfer of pressure, and there is no such undue concentration as in Fig. 265. This disposition of the timber affords

FIG. 266.

a greater strength, and is the leading feature. Additional braces are sometimes added and the sets connected every which way; but the design is to arrange the timber of each section in such manner as to form an integral part of the completed system. After the roof-timbers are in place, plenty of space is left below for masons, and good ventilation is had.

Comparison of the Four Systems.—Each system has distinctive features commending it to favor under given conditions, and a brief comparison may be given.

The centre-core system is emphatically inapplicable for loose rock where other methods are better. It is quite expensive in all of its operations, and is not safe.

The English system is safe enough for most tunnels that are likely to be driven, and offers ample space for the miners and masons to work, but it is not adapted to very bad ground or where heavy pressure is encountered.

The Belgian system seems to be preferred for moderately firm ground, in which it is quicker than either of the two above mentioned; but it is hardly suited to running ground, because of the great care which must be taken with the underpinning. The cost of transportation while building is higher than in the others because of the frequent interruptions caused by the transportation scaffolding. In very loose ground it is positively dangerous.

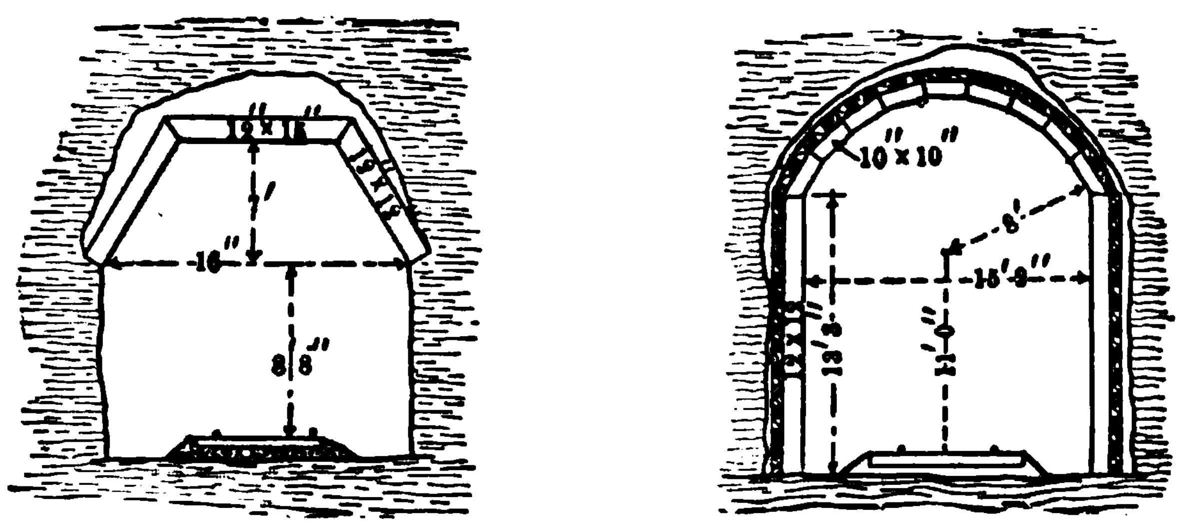
The Austrian system has more to commend it and less to condemn it than the others. All the driving operations are cheaply and easily conducted. The driving is least interrupted, and safety is assured if the middle timbering and the longitudinal bracing be well apportioned. The forepoling, if employed, would be in the direction of the tunnel axis, and the masonry arching is not delayed.

Block Timbering.—In Figs. 266 and 268 are illustrated the timbering permanently placed in the tunnel. Each of the nine blocks is 10″×10″, the legs being 12 inches square. From the three-rafter system of roofing the tunnel American engineers developed the block-timber arching for the support of the roof rock which is very loose and requires close bracing. The wooden voussoirs were first used for temporary support only, until they were replaced by brick or stone arching, but latterly they have served as a permanent arching lined with a fire-proof sheeting for railroad use.

We cannot claim any system of tunnelling as our own, for

neither the number of tunnels nor the difficulties encountered are as great as in the Old World. The Austrian method is the nearest approach to ours, or, rather, is the one which our engineers have adopted with a modified framing. The mode of driving and timbering is illustrated in Fig. 264. The upper heading, 1, with its enlargement, 2, 2, precedes the work upon the "bench." With a tunnel area of 21″×27″ the heading is about 8 feet high, and the bench, the remainder, attacked in one section. The upper heading is timbered, rafter style, three or nine voussoir bents forming a block-arch on heavy permanent pole standards, or on the side rock if sufficiently sound. In loose rock the number of bents increase and the timbering is heavier. Inside of this frame arch the masonry is built. A segmental arch 21 feet high by 18 wide in the clear had nine voussoir-blocks 25 inches long and 10″×10″ section; two wall-plates, 6 feet 4 inches long, 12 inches square; two posts, 14′ 1″×10″×12″; and 45 lagging, 6′ 6″×6″.

Fig. 268 illustrates the style of bar-timbering, which, however, will not permanently withstand much lateral pressure.



FIGS. 267 and 268 —The Block Arching of Tunnels.

Tunnelling Soft Ground.—In running ground the area attacked is limited by the rapidity with which permanent support can follow the excavation, during which provision must also be made against the pressure coming from all sides. In no region, not even in the Rocky Mountains, is the engineer free from the liability of striking ground that may overwhelm the miner before timbers can be inserted; so that the ground has to be restrained up to the face as well as behind the frames, with which close

lagging may suffice to prevent movement. If it is not in new ground, the consistency of which will determine the details of its penetration, it may be where timbers have rotted or given way that a cave or a run may occur.

In alluvial or sand the method of spilling or a pneumatic system is indispensable. These have been employed under the waters of the Thames, the Severn, and the Mersey Rivers in Great Britain and are universally employed for driving through similar ground in America.

Spilling by laths is a method equally applicable to drifts as to shafts. In front of one stick of a set, and behind its mate in the next advancing set, pointed heavy planks are driven, one set in advance of the face, close together on one or all of the sides from which the pressure is exerted. The fore end of the plank is forced down upon its set, while the rear end is held against the lower side of its cap, being protected from pressure by the previously driven upper lath. This is the method of forepoling illustrated in Fig. 269 The progress in soft, heavy timbered rock is three times as fast as in very solid rock.

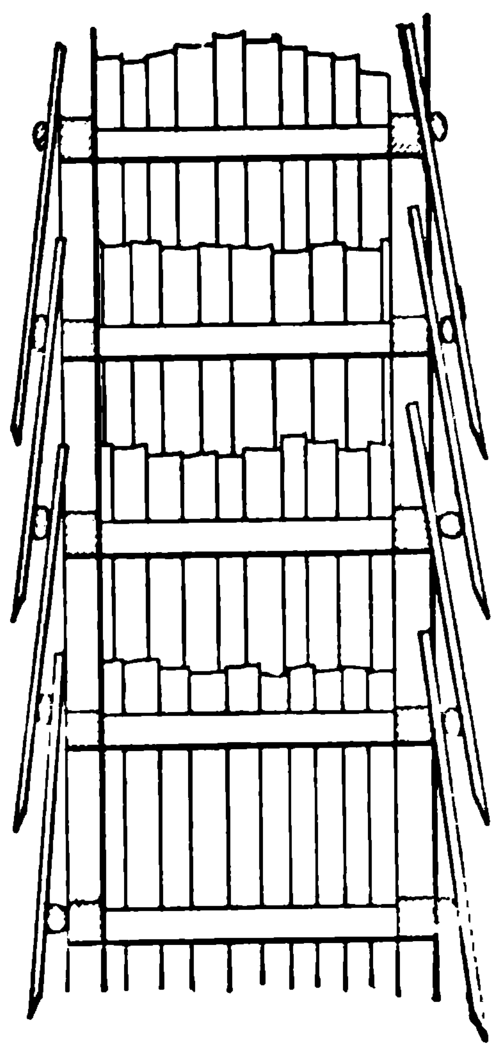


FIG. 269.—Forepoling.

The "spilling" protects the fore-breast by horizontal laths, as long as the breast is wide, held against it and braced to the nearest set. Each lath, *a* (Fig. 270), is removed in descending order to permit some ground to run off; it is advanced a short distance and braced again, *b*. The progress depends upon the speed with which the spaces may be opened and closed; as these are small, the movement is controllable, and there need be no fear of sudden shock to the timbers. The method is simple, and has been eminently successful under many circumstances; once, with masonry lining following the forepole, a tunnel was executed within 18 feet of a river-bed.

FIG. 270.—Method of Spilling for Running Ground.

An entirely different principle is that employed by Durieux and others in Westphalia, whereby the ground was forced ahead of the work and was not removed at all (Fig. 271). The walls

FIG. 271.—A Picketing Method for Sandy Ground.

and roof were forepoled, but the breast and floor were checkered by pyramidal pickets, completely covering the exposure. Those on the face were square faced and larger than the floor-wedges, which were 12 inches long and 4 inches diameter, driven by mallets. The floor-pickets remain in permanently, while those at the face force the soft material ahead and advance in this manner. The battering they receive renders them useless in a very short while, when they are replaced. There is no material to be hauled except that used in the construction, and rather bad ground has been thus traversed. On occasion a short lateral drift is similarly pushed to relieve the main work. Four men in a drift 5′×6′ in morainal matter will advance 4 feet a shift.

The finest piece of tunnelling is the construction of the new Croton Aqueduct, where water, mud, quicksand, and all varieties of loose soil were penetrated in an area of 676 sq. ft. Forepoling for the roof and sides, picket-spilling for the floor, and the American system with block-arching were adopted at various

FIG. 272.—The Brunel Shield.

points. The ground in extended places was so bad that 24-inch timbers were crushed by the pressure.

Iron Shields at the Tunnel Face.—Iron is a safer constructive material for loose soil; and in 1825 Brunel put it to use for excavating an area 38×22 under the Thames River. As in the spilling method, the face was covered by laths about 3″×3″×6″, and for rapidity of work was subdivided into three tiers of 11 breasts each (Fig. 272), each being protected by a cast-iron shield, from which struts hold the laths *d* in place. One man to a cell operates by taking away a lath and replacing it 3 inches in advance. The laths are removed successively downwards until 3 inches of progress has been made, after which the alternate frames are carried ahead 6 inches and the performance repeated. In each cell is a similar performance. The masons follow the miners very closely at *G*.

The second Thames Tunnel was completed by the use of a shield, which with pointed shoes was forced into a stiff clay at the rate of 9 feet per day by jack-screws exerting a pressure of 60 tons on it. Three men crawled through a door in its face and excavated some earth preparatory to the next move.

The Hydraulic Shield.—The present practice for subaqueous tunnels involves some variation of the pneumatic system. A pilot-tube or caisson penetrates the soil, which is held up by compressed air. The masonry, perforce, is built as fast as the shield, tube, or caisson advances. In the Greathead system a cylindrical iron shield, 21 feet in diameter, is thrust from the masonry by hydraulic pumps under a pressure of 3000 lbs. per square inch. The front edge of the shield is a heavy ring shoe to facilitate its advance into the silt. The rear end of the shield encloses the masonry, which follows the progress of the shield. Doors in its face are opened to allow the soft earth to enter. This is promptly removed (Fig. 273).

In preparing to tunnel silt, both the weight and the vertical pressure of the overlying material and the lateral movement of the loose paste are to be resisted. The first is a matter of determination, and the ability of the completed structure to

FIG. 273.—The Hydraulic Shield.

withstand this is also a matter of mathematical calculation; but the second is the difficulty to be apprehended. The Hudson River Tunnel engineers, however, satisfied themselves that the tendency of the gravel to pour over the top of the tube would reduce the lateral stress to the resistance of the tube. Certainly their experience in subaqueous driving goes to show that the tube serves better than the timbering systems as regards the prevention of overhead settlement. The liability to settlement in front of the shield may be overcome by grouting under pressure at the rear of the shield between the lining and the roof.

The Anderson Pilot-tube.—This is a segmental tube of about 6 feet diameter, strengthened inside by radial timbers, which precede the main work. It is of $\frac{1}{4}$-inch plates, $12'' \times 24''$, riveted together by means of flanges; and when a cut has been excavated into the heading large enough, one of the plates is placed and

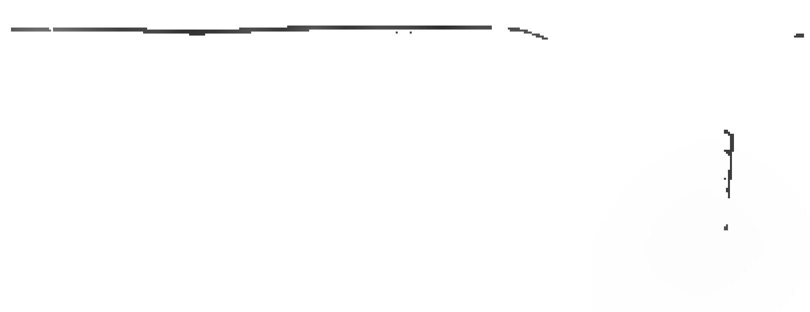


FIG. 274.—The Anderson Pilot-tube.

held by props (often the plates are held by compressed air during the work); on each side other cuts are made for two more plates, which are riveted to it. Rings of the pilot are thus successively completed.

Around this, in small terraces, and considerably behind the pilot, the main shell, 17 feet in diameter, is finished in a similar manner, the plates being propped from the pilot-tube, which is always braced from the masonry that lines the shell. With its progress the rear rings of the pilot tube are removed and their plates shifted to the front end. The masonry consists of six courses of brick laid in cement. To reduce the volume of the

tunnel that is kept under the compressed air, brick bulkheads, 4 feet thick, provided with two air-locks, are built every 400 or 500 feet. Only the two nearest the work are maintained.

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CHAPTER V.

DRILLING- AND BORING-MACHINES FOR EXPLORATIONS.

Classes of Machines.—The penetration of rock or of loose material may be accomplished by either of two operations represented by corresponding types of machines. By a repeated reciprocating blow or concussion upon the rock, fragments may be broken therefrom and a hole drilled to any depth and at any rate, depending upon the strength of the machine and the resistance of the rock to abrasion. Machines operating upon the rock in this way are known as percussive machines or punch-drills. The second method of penetration consists in employing a rotary cutter which, with sufficient pressure or weight upon it, will abrade the rock at a rate depending upon its relative softness and the motor power.

Percussive drills are employed to advantage upon hard uniform rock only. Rotary borers are constructed of various types as to be equally applicable to either soft rock, loose ground, or hard rock.

These drills or borers may be operated by directly connected engines upon the same frame to drill holes of thousands of feet in depth for prospecting purposes. When these machines are to be used underground they are of a more portable type and operated by electricity or compressed air, conveyed to it from a distinct source. The depth of the holes made possible with these is limited to a few feet. Hence their utility is confined to drilling holes for blasting purposes and not for exploration.

Prospecting-machines.—The machines employed for prospecting are either of the punch-drill type represented by the American oil-well rig, or the rotary types represented by the "diamond" drill and other forms of tubular drills.

Bore-holes are driven for the testing of the strata; to prospect for the gas; to afford an outlet for water; to pump brine to the surface; or to subserve some precautionary measures in mining, such as special ventilating openings, deep sump-holes to drain the mine into a lower porous stratum or conduits, for tail-haulage rope. These machines are also employed for drilling a series of holes in connection with the Poetsch method and the long hole process for shaft-sinking.

Percussive Drills.—Reciprocating motion is obtained from a steam- or air-piston either directly attached to the drill or indirectly connected with it by rope. The blow may be produced by the pressure of the motor fluid in the first case or by the weight of the drill allowed to fall freely after having been raised by the rope. In the first method the piston pressure is great compared with the weight of the tool, while in the latter the force of the concussion is solely due to the heavy weight of the drill. The early type of punch-drill consisted of a continuous line of iron rods, lengthened from time to time with the progress of the hole, which was raised and lowered at the surface by a walking-beam operated by a cam or other lifting device. The shock transmitted through the long line of metal caused breakage of the tools, and other difficulties which led to the later forms of machines in which heavy drill lengths were suspended from the surface by a flat or round rope and at a much more rapid rate than the line of rods. Later the solid rods were replaced by several lengths of hollow rods which gave greater progress with the same power, and overcame the difficulties of feeding water and disposing of the slimes produced during the drilling. The rods were screwed together or bayonet-hooked, the former being preferred. The engine may be single-acting with a reversible link motion provided with a sectional fly-wheel, whose weight was increased as the hole deepened. It communicates motion to a walking-

beam which is pivoted at its centre on top of two sampson-posts and fastened at the derrick end to the drill-rope attachments.

Oil-well Rig.—The typical American drill rig is illustrated in Fig. 275.

This consists of a derrick 75 feet, or more, in height, having at its crown a sheave over which the drill-rope passes to the "bull-wheel," or large drum. A high derrick is required to facilitate

FIG. 275.—An American Drill Rig.

culties as were sough to be removed by various appliances. Oennhausen's chisel had at the top a four-armed projection which played freely in corresponding slots of a cylinder attached at one end of the rope. As the rope was lowered with its drill it was turned slightly and the ledges of the cylinder slipped

under the cross-arms of the chisel. This was then caught on the up-stroke of the rope and its cylinder. When it was raised to the proper height a slight jar or jerk released the tool, whose cross-arms slid in the slots and permitted it to drop a distance equal to the length of the slots. The invention of M. Kind and Chandron consisted of a pair of bent levers acting like pincers which grasped the tool and raised it to the proper height, after which it was allowed a free fall to do its work without recoil upon the apparatus.

The jar universally used is illustrated in Fig. 276. It is a pair of open links allowing of 1 to 3 feet of play. One link is attached to the upper rod length and its rope, the lower to the drill-bit. These aggregate 1200 pounds in weight for an 8″ hole and a total drilling string of 60 feet.

The Drill-tools.—In Fig. 276 is illustrated a full string of tools. Beginning at the left they are: the sinker-bar, wrench-bar, wrench, jars, temper-screw, drill-bit, spudding-bit and rope-socket, and auger-stem. Above is the floor-circle.

The temper-screw is suspended from the walking-beam and regulates the feed of the drill and the rope. The swivel below is turned a half-revolution after each blow, paying out the rope and rotating the drill-bit a small amount. When the full speed of 4 feet has been paid out the screw is unclamped, while the drill-rope is otherwise supported, and its feed raised for the next stage or drilling.

The stem, drill-rods, jars, and bit are about $1\frac{1}{2}''$ in diameter, weighing, for a 60-foot sring, 1200 pounds. Each member has a screw-pin at its upper end, fitting accurately into the box at the lower end, leaving the outside of the flush. A pair of wrenches is employed for tightening, one below and the other above the joint. The upper one is further aided by the wrench-bar inserted into a hole in the floor-circle bolted to the frame.

Drilling the Hole.—The hole is started by the spudding-bit and stem, which are attached to the rope passing over the sheave and once or twice around the bull-wheel. The engine in opera-

FIG. 276.—A String of Drill-tools.

tion revolves the bull-wheel continuously. The free end of the rope being held by a man, the bit and rope are raised a slight distance, after which the rope is released, slipping on the drum, and allows the tool to fall. This is repeated till the surface-soil has been penetrated. The verticality of the hole is maintained meanwhile by an upright plank box leader. Formerly the tool was manipulated by a spring-pole with a foot-stirrup for a moderate depth and was known as "kicking down a hole."

When bed rock has been reached the spudding-bit is replaced by the drill-bit, stem and jars, and their rope clamped to the temper-screw which communicates the reciprocating motion from the walking-beam. The tool is raised from 2 to 6 feet and allowed to fall at a rate varying with circumstances, the average being six times a minute. After each blow the swivel is turned, for a uniform rotation of the drill is essential and a constant watchfulness necessary, otherwise the drill may work out of line. This is particularly essential if the strata are inclined or the material not homogeneous.

Frequently, to maintain verticality and sometimes to straighten a curved hole, a winged tool is brought into play. The drill-rod is provided with projections extending on the four sides of the chisel for the same distance along it and nearly fitting the hole, and sometimes a hollow cylindrical reamer is resorted to for the purpose.

In the Mather and Platt system the partial revolution of the tool and its flat rope was obtained after each blow by a movable collar rotating with the tool. It was cut with inclined teeth at both ends and was capable of a motion of 2″ or 3″ vertically inside of an iron bow. Above and below the collar were sets of inclined teeth into which the collar ends geared. With each drop the collar engaged the teeth of the lower set, turning it with its chisel one half a tooth. At the end of the upper stroke the collar strikes the upper set and is again turned one-half of a tooth.

When the temper-screw has reached its limit it is returned to the starting-point, fresh tools meanwhile being attached; or the débris is pumped out preparatory to the next run. If no under-

ground stream has been encountered to supply water for cooling the bits and facilitating the removal of the débris, it must be furnished at the surface.

The Sand-pump.—This consists of a slender cylinder with a valve at its foot which is attached to a rope having its own pulley at the derrick head, and its own wheel at the engine. It is lowered into the hole after each run and pumped up and down. The sludge produced by the drill and the water is drawn into the pump and raised to the surface.

Recovering Lost Tools.—If the sand-pump or reamer breaks loose, it can be fished out by some form of grapnel, or it is chopped up. If pebbles fall into the hole above the tool and wedge it fast, it may be jarred loose or freed by a spear. Occasionally the hole is reamed out to a larger size.

The Progress.—The rate of progress varies from 10 feet a day in magnesian limestone to 70 feet in soft ground. Much depends on the skill of the driller, who may determine by the "feel" of the concussion transmitted through the rope whether the blow has been effected or is cushioned. The feed is regulated accordingly. Two men are sufficient to operate a drill rig, and 200 pounds fuel are consumed per day.

The cost of drilling wells is 50 and 60 cents per foot, though some very deep holes average $1.14. Reaming is done at a cost of about 40 cents; and the royalty for the rent of a machine is nearly 10 cents.

Hand-boring of Deep Holes.—A unique mode of drilling artesian wells is in vogue in San Luis Valley, Col., where 300 feet of 5″ hole can be made in twenty-four hours. The soil of that valley is very porous, and for 900 feet down will hardly hold water except in the clay seams. Ditches have failed of their irrigating purposes, and each ranch is provided with one or more of these spouting wells. From a tripod a 5″ tube is held vertically and by a pair of blacksmith's tongs is turned by hand. At the foot of the pipe a bit projects outward ½ inch or so. By pressure and rotation a hole is cut spirally into the gravel, lengths being added with the progress. A barrel or two of water is

poured down the pipe to wash the detritus out through the annular space until a ground-current is encountered which will supply the necessary water. Beyond 300 feet horse- or engine-power will be required to turn the borer.

Casing a Bore-hole.—When a loose or fragile stratum is encountered its caving interferes with the progress of the drill. A pipe-tubing is forced down to support the sides of the bore. It may be driven while the drilling is in progress below or the operations of drilling and casing may alternate. Though wood is the best tubing material, the objections to it are so apparent that wrought-iron pipe is used. The pipes may be screwed together, joined by wrought-iron couplings or telescoped. The first length, having been sharpened, is driven by percussion until its full length is in the guide tube and the successive lengths are screwed on and hammered. When the depth has so far increased that the pipe can no longer be rammed, the bore-hole is enlarged for the same size of pipe to be lowered, or the same diameter of bore is maintained for a smaller pipe. The pipe is driven by repeated blows from a drop-weight or by the steady pressure from jack-screws, a block of wood at the top of the pipe-line receiving the impact.

A deep bore-hole usually encounters water-currents at different depths before the object for which the bore has been drilled is attained. These are isolated by casing. A salt-water current may be met before the artesian flow is reached or either of these may be intercepted before the oil-sands are struck. A pipe of large diameter may be carried to the first flow and sealed at its foot. A second smaller pipe, lowered to the next flow and sealed, would afford an escape for the upper current in the annular space and a discharge for the lower one inside the smaller pipe. Three separate artesian flows are thus obtained by casing each current and discharging them at different surface elevations. An undesirable current may be dammed back in like manner. This may be necessary when drilling for oil, as the accumulation of the flowing water reduces the efficacy of the blow and also exerts such pressure, at the rate of 14.7 lbs. per square inch for every

34 feet in depth, that when the oil stratum is reached neither oil nor gas can escape.

The casing is sealed by hydraulic cement or by the use of a seed-bag. A stout leather tube several feet long is attached outside of the piping at the lower end. The annular space is filled with flaxseed bran or other absorbent material and the top lashed. When lowered into place the seeds swell and furnish a tight joint somewhat like that of the moss-box (Fig. 203).

FIG. 277.—The Cyclone Prospecting-machine.

Shooting the Well.—It frequently occurs that the oil does not flow freely, or even at all, when the sands are encountered. A practice prevails of opening communication with an underground oil-chamber by exploding a torpedo at the bottom of the well. This is a charge of nitroglycerine carefully lowered into position, after which a heavy weight is dropped upon it. This heroic treatment is not resorted to in shaly ground, unless provision is made for immediately casing the shattered portion of the hole, as it often results disastrously. The charge varies from 10 lbs. to 100 lbs. in desperate cases.

Removing the Casing.—When the object for which the hole was drilled or lined has been attained the tubing may be recovered by the use of an ovoid screw-plug of oak, which is attached to the tool rod and lowered to the bottom. A little sand is poured in and wedges the plug, which can be hoisted without trouble. To loosen its hold on the tube it is only necessary to drop it slightly and let the sand run out. A three-

FIG. 278. FIG. 279. FIG. 280.

FIG. 278.—The Keystone Sand-pump.
FIG. 279.—Casing a Well in the Keystone Process.
FIG. 280.—Keystone Pipe-puller.

pronged expansible hook lowered down under the tubing is also called into service.

The Keystone Prospecting-machine, shown in Fig. 279, is a type of percussion drill, giving service in loose placer-ground and the zinc-lead fields of Missouri. The string of tools is identical in character with those of the oil-well rig. The hole is started with a fluted spudding-bit, stem, and rope-socket and alternated with the sand-pump (Fig. 278). When 50 feet has been drilled the jars are inserted in the line and the hole continued.

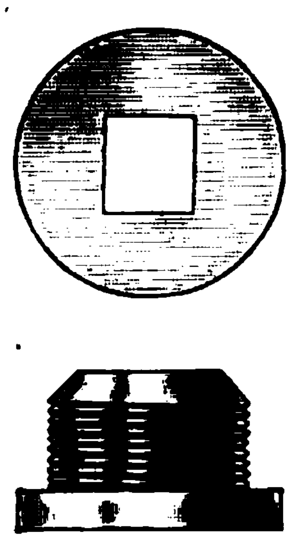


FIG. 281.—Keystone Pipe-puller.

The drive-pipe casing is of merchant pipe, with straight threads and squared ends, in lengths of 6 feet, coupled together. At the foot is a heavy drive-shoe with a cutting-edge and at the top a cap on which the driving blows are delivered. The casing follows the drilling closely as shown in Fig. 279.

The drive-pipe is withdrawn by the pipe-pullers (Figs. 280 and 281).

The Cyclone Drilling-machine resembles the other portable drill-rigs, though the detail of operation is somewhat different. The drill-bit is not solid but is hollow-stemmed. Holes in the side of the bit afford communication from inside to outside. Hollow rods support the drill-bit instead of a rope and at their junction is a ball valve. When the line of tools falls, the valve opens and permits the drillings to enter the drill-stem and rise in the hollow rods. On the up stroke it closes. With each down stroke additional material enters, to be finally discharged at the surface into a proper receptacle. Extra-heavy rod lengths are employed near the bottom and lighter rods above. For holes exceeding 4 inches in diameter the rods are of 2-inch and 1½-inch pipe (Fig. 277).

Rotary Drills.—M. Leschot has the credit for the first application of rotary diamond-drills to the miner's art, since which time it has gained in favor and increased in range of utility. The lower end of a long line of tubing is supplied with a cutting-

surface and is caused to rotate by appropriate machinery at the surface. A continuous pressure is produced at the cutting-edge by the weight of tool above. The cutting-medium may be chilled-steel fragments, diamonds inserted into the rim of the boring-tool or a steel tube may be dressed to a hard cutting-edge for the purpose, each having its special adaptability to rock of a definite character.

The Diamond-bit.—A cutting-tool is constructed of diamonds which are forced by hydraulic pressure into sockets on the end of a steel tube. Recesses are accurately prepared into which they are secured by metal hammered up around them. In some cases a firm setting is obtained by forcing the stones forward through small holes in the metal by means of a screw or by hydraulic pressure. A later method consists in forcing the stones nearly through the metal and subsequently grinding the steel down until the stones are exposed.

The tube may be closed by a concave surface (Fig. 284) or by a convex face (Fig. 283) and the entire area studded in such manner that no concentric circle can be drawn failing to touch one or more of the diamonds. Some also project beyond the

FIG. 282. FIG. 283. FIG. 284.

The Annular, Convex, and Concave Diamond-drill Bits.

tube. The annular end of a tube may be charged with diamonds, as in Fig. 284. For drilling a small hole the first-named types of bit are used, the concave being preferred to the convex, but for large holes or where a central core is sought and withdrawn for inspection the annular bit is used.

The diamonds are of two varieties, the black "carbonadors" and the deep-red "borts." The former lack the crystallization

making them durable, while the borts are rather brittle. Theoretically too many carbons cannot be put in; "there should be never less than 12, and as many as 20 may be mounted on a bit."

The Diamond-drill.—The bit is coupled to a line of tubing and guided by a short or long length of tube (Figs. 285 and 286). The tubes are in 8-foot lengths and of an outer diameter of rarely over 8 inches. Holes of great depths are thus drilled of a uniform diameter to the bottom or in sections of diminishing diameters.

Fig. 285.—The Guide-bit.

The engine power varies with the length and size of drill-tube to be manipulated. For 1000 feet of bore an 8-horse-power engine will suffice. This with its running gear is framed and suitably braced to carry the full line of rods. A derrick is often erected

Fig. 286.—The Core-barrel.

to facilitate the addition or disjointing of tubes.

Regulating the Feed of the Drill.—The changes in the structure and toughness of the rock during drilling are so rapid and pronounced that the drill cannot be fed uniformly. A variable feed is necessary commensurate with the progress of the drill and two methods of avoiding a positive feed are in vogue. One is a spur-wheel feed, the other an hydraulic. The former is so adjusted by differential gear that its friction shall equal a desired resistance, and when this is exceeded because of undue strain below, a regulation is obtained. This is inferior to the hydraulic feed. In Fig. 288 is shown a simple motor which by means of hydraulic pressure on the piston produces a pressure which is maintained constant. Both ends of the cylinder are connected with the pump and the suitable cocks admit of a per-

fect control by the operator, who gives any variation or reversal of speed within the limit of the pump and the piston area. When the pump pressure increases, the piston *B* and its rods

FIG. 287.—A Prospecting Diamond-drill

are raised. A decrease of pressure results in its lowering. The rate of feed is fixed by the hardness of the rock. In Fig. 287 the feed cylinder is shown in position above the drill-tubes. The pressure exerted by the feed is just sufficient to produce abrasion, but not to cut the rock.

A chuck intervenes between the tube and the feed, which may be loosened at the end of the feed-travel and run up to the top for a fresh start. The tube line is partially suspended by friction-rollers at the surface, so that it is subjected to very little tension. But a high degree of torsion is developed by the rotary effort against the abrasive resistance, and the torsional strength of the tubes places a limit on the possible depth of drilling. The regulating power of the feed limits the capacity of the machine.

Removing the Cuttings and the Core.—The tube is revolved at a rate of from 400 to 800 turns per minute and continues without interruption from five to eight hours, cutting meanwhile from 13″ to 2′ each hour. A stream of water is fed inside of the pipes and washes the rock face and the annular bit. It carries the cuttings up through the annular space outside. The grooves on spiral the outside of the pipe (Fig. 286) facilitate the escape of the water with the solid bits. The water is fed outside the pipes and escapes with the débris through holes into the interior. A guide-bit (Figs. 285 and 286) and core-barrel maintain the bit in the direction in which it is started. The water-supply is forced by means of a pump.

FIG 288.—The Hydraulic Feed.

The progress of the drill is carefully

watched and the character of the cuttings supplied by the overflowing water-current is examined. Samples are taken and preserved for reference. With a solid bit is produced only a fine sludge for inspection, but the annular bit furnishes in addition a core.

With every 10 to 15 feet of advance the tubes are raised for examination and for the purpose of extracting the core. In the inside of the drill-tube is a lifter (Fig. 289). This is a spring ring which grasps the core only when it is raised It wrenches the latter from its place, and will retain hold on it until released. This can be done by lowering the tube.

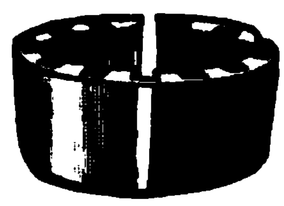


FIG. 289.—The Core-lifter.

The Core.—The core furnishes a guide also as to the direction of inclination of the strata if care be taken that the tubes are not turned while being lifted. Flat ropes are therefore used for hoisting as less likely to untwist and turn the drills. False information may be given by the core if a soft layer has been penetrated between two hard strata, for the former may wear away or even grind off loose in the core. Hence the core may not be an absolute guide. Many causes combine to make this examination unreliable. A careful measurement, an allowance for wear, and frequent inspections are the only means of checking results. The progress in homogeneous rocks is uniform, but a composite rock with constituents of unequal hardness is difficult to penetrate in perfect alignment. In clay a thick pulp is formed which chokes the tubes and requires the full pressure of the pump to remove it.

Deflection of the Drill-hole.—Changes in the inclination of the strata or variation in hardness of the material tend to deflect the drill-tubes and to buckle the rods under torsion. The correctness of the alignment must be maintained, and this may be tested by a phial of gelatine lowered to a certain depth and allowed to harden. When raised to the surface, it is inclined in the same plane it occupied till the gelatine level is horizontal, whence the dip of the hole is made known. The line etched by hydro-

fluoric acid on glass is another method of test. Neither of these are entirely satisfactory.

Accidents are Rare.—Occasionally a diamond may fall out, and if it cannot be recovered, is chopped up at once, or the direc-

FIG. 290.—An Underground Diamond Drill.

tion of the water feed is reversed and its pressure increased to wash the stone up the tube. Hard nodules of rock that retard a fair progress are chopped up by a special bit. A diamond has also been recovered by inserting a lump of wax at the bottom of a tube, which is lowered into the hole.

The limiting depth of vertical holes is determined by the torsonial strength of the tubes and the power of the machine. That of inclined holes depends upon the amount of friction between the tube and the rock.

For underground work a three horse-power electric motor like the "Little Beauty" (Fig. 290) is mounted on a truck, with

FIG. 291.—Exploration by Diamond-drill Holes.

drum-drill and a pump, and permits core-drilling to advantage in small spaces. In many mines 1-inch cores in sections of 5 to 20 inches are cut for 80 feet depth, and a great deal of prospecting has been done with this compact machine, which often makes 2 feet per hour at a cost of 68 cents to $1.06 per foot. In Fig. 291 are exhibited the explorations conducted at the Silver Islet Mine by the use of the underground drill. Doubtless many properties owe their existence to the result of diamond-drill discoveries, and its use has frequently saved expense in various ways. By graphically representing on cross-section paper to

scale the results of underground borings a more intelligent interpretation can be made. On the charts are indicated in colors the various rocks penetrated by the several bore-holes. A comparison of position and thickness enables the engineer to judge of the prospects. Still, the diamond drill is not considered infallible in its indications as to the presence or absence of the ore body sought.

The Chapman hydraulic pipe-rotating drill and the Davis calyx drill are types of rotary borers serviceable for loose ground such as that of the Beaumont oil-field of Texas. Both systems have a turntable for directing the drills, and a pump for the water-supply. The Chapman drill depends upon "adamantine" for its abrasive material, and the Davis drill uses chilled shot in hard uniform rock, but the normal bit in soft ground. The drillings pass up the tubes and are caught in a calyx above the Davis core-barrel, or at the surface in the Chapman system. A rate of speed has frequently been attained equal to 900 feet in twenty-five hours. In California an experimental hole was drilled 1500 feet in seven full working days. Holes of 15 inches diameter have been bored by these systems and to a depth of 3000 feet.

CHAPTER VI.

MINERS' TOOLS.

The Texture of Rocks.—Rocks to be excavated or broken include those which are hard or soft, tough or brittle, and creviced, stratified or massive; and the difficulties of their removal present such varied and delicate questions that it is difficult to give a systematic account of the principles of breaking ground or the appliances employed. Materials are gauged by their resistance to abrasion being called "hard" or "soft" as they affect drilling operations; and *tough* or *brittle* according to their resistance to concussion in blasting. These qualities determine the cost of breaking ground. For example, quartz is hard wherever encountered, but not all of it is tough. Hard minerals wear the cutting tools rapidly, but a brittle mineral is easily shattered by the blow of a hammer or the concussion of a drill. A tough rock possesses the tenacity which resists the rupturing agency of an explosive. It will require powerful blasting agents for its removal. Porphyry is of an average degree of consistency and may be easily drilled and readily broken. Metamorphic rocks are frequently hard, but always tough. They drill easily, but do not break freely. Trap-rock, syenite, and granite are hard on the drills, but require little explosive, the grains having slight cohesion. Fire-clay is tough, though very soft. It has an elasticity which prevents much impression being made on it by an explosive.

The massive rocks are usually homogeneous and present the same resistance throughout their mass. Some stratified rocks and gravels are not, being composed of hard grains or pebbles cemented by a softer material. These are easily ruptured by powder, but are difficult to drill into. Creviced rocks are not

economically broken by explosives, but are split by wedges and slow rupturing agents, utilizing the weak lines of cleavage. Bituminous coal affords examples of a rock split up by numerous planes into prismatic fragments which would permit the gases from powder explosions to escape. Anthracite being brittle, of short texture and hard, compared with bituminous coal, therefore requires powder, while bituminous coal is mined by picks.

The engineer must therefore be familiar with the texture and behavior of the rock before any estimate of the cost of its extraction can be made. In Drinker's "Tunnelling" and Foster's Callon's "Lectures on Mining" are given elaborate formulæ for estimating the cost of extraction, and they may serve as approximate guides. In each camp a close observation of the results of experiments will supply the practical coefficients which can be used in the above-mentioned formulæ for determining the cost of working.

Fire-setting System.—This is a method applied by the miners of the middle ages to conquer hard rock by the instrumentality of fire. The face of the rock was exposed to the heat of fire and then suddenly cooled by water. The unequal contraction, upon cooling, softened the rock, which then was amenable to the ordinary tools. A grate was built inclined toward the face of the rock, and in it were piled billets of wood. When ignited, the flame was directed against the breast by a shield overhead. In some cases a basket was suspended from the roof with the same result. Usually in the mines of the early times the wood was fired on Saturday and allowed to burn out. The men could attack the calcined surface upon their return to work. This method may profitably be applied upon hard rock where fuel is cheap and the ventilation is good.

Miners' Tools.—The operations to be performed by the miner consist in breaking down the mineral or rock, removing the broken material, and illuminating the place of work. These require groups of tools the nature of which varies with the character of the mine in which the work is performed. The mineral may be broken down by picks, drills, striking-hammers, sledges,

augers, gads, moils, wedges, and feathers. Auxiliary tools are also employed, such as the swab, spoon, gun, etc., when the material is tough enough to require blasting. In the latter case, in addition to the striking-hammers and drills, will also be required the tools for making powder cartridges, cans for carrying the same, and needles, squibs, fuses, etc., to ignite the powder. The mineral is removed by the aid of shovels, picks, and spades; occasionally sledges and crowbars are required for loosening or breaking fragments too large for convenient handling. The illumination of the place of work will require a candle and its stick, or oil-lamp and an oil supply.

All tools loaned or handed to the miners are entered on an account against them, as otherwise many would be lost in the waste, or otherwise disappear. A check is made of the condition of all tools returned by the men. Steel should be measured or weighed out on delivery at the beginning of the day, month, or contract.

Rotary Drills.—The second class of drills cut away the rock by the rotary motion, and according to the character of the cutting bit they are breast-augers, roof-augers, and machine-power augers, worked by compressed air or electricity.

Hand Boring-machines consist of a drill-piece of twisted steel, welded to an iron stem and fixed to the end of a long screw passing through a nut which is supported on an upright bar by a ball-and-socket or universal joint, permitting any direction of bore. The rotary motion is communicated to the drill-screw by two bevel-gear wheels, or directly by a revolving handle at the rear of the screw. The bit may be a wedge-shaped bit with a V edge, or it may be twisted into the common auger form.

The form illustrated in Fig. 292 is a special bit used by bituminous miners. The shank to which the twisted steel bit is welded is bent to form a handle and terminates in a breastplate (Fig. 293). The boring of the holes is accomplished by hand, the pressure being applied by the weight of the body against the plate. The bits of these augers are from 1½ to 2½ inches in diameter. Sectional augers are more convenient for delivery to the blacksmith, therefore they are made removable from the shank.

Of the hand- and roof-power augers there are several styles, the two general classes being the grip-drill and the post-drill. The grip-drill (Fig. 294) has a metal rest *a* for the feed-screw nut *b*, in which the feed-screw *c* turns while the hole is being bored. A small recess in the coal is provided into which the grip-bar *d* is driven and wedged and thus gives stability to the machine. The boring can be done from the side or end.

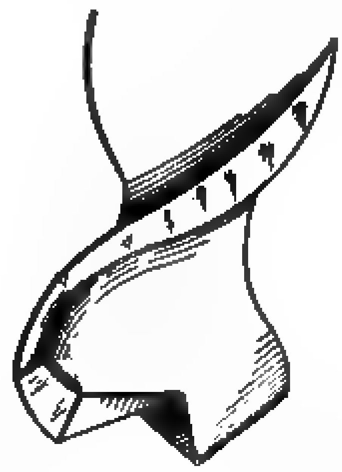


FIG. 292.—A Bituminous-coal Bit.

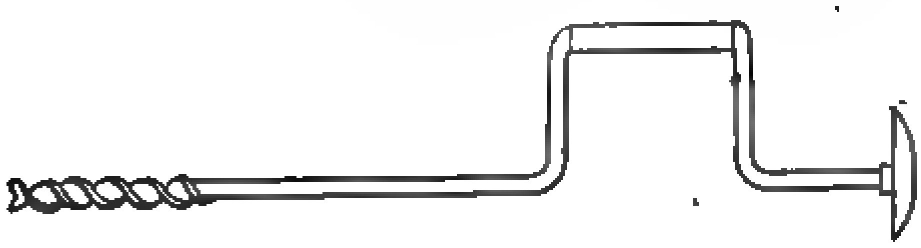


FIG. 293.—A Breast Drill.

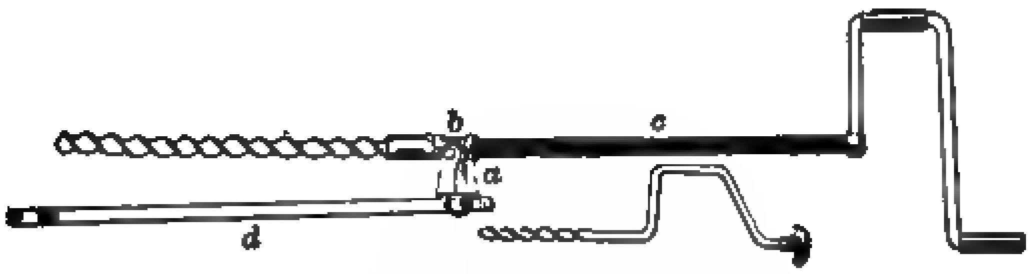


FIG. 294.—A Grip-drill.

The post-drill consists of a suitable borer attached to the post, which is usually placed upright, having a jack-screw at one end and firmly wedged between the floor and the roof. The post is split or may be obtained solid. The drill is attached to the feed-screw which advances through the feed end by a handle and crank operating the gearing shown. The universal joint supporting the drill-gear permits holes to be drilled in any direction.

Portable Power Borers.—The Jeffrey rotary electric coal-drill (Fig. 295) has a small enclosed motor on the end of whose

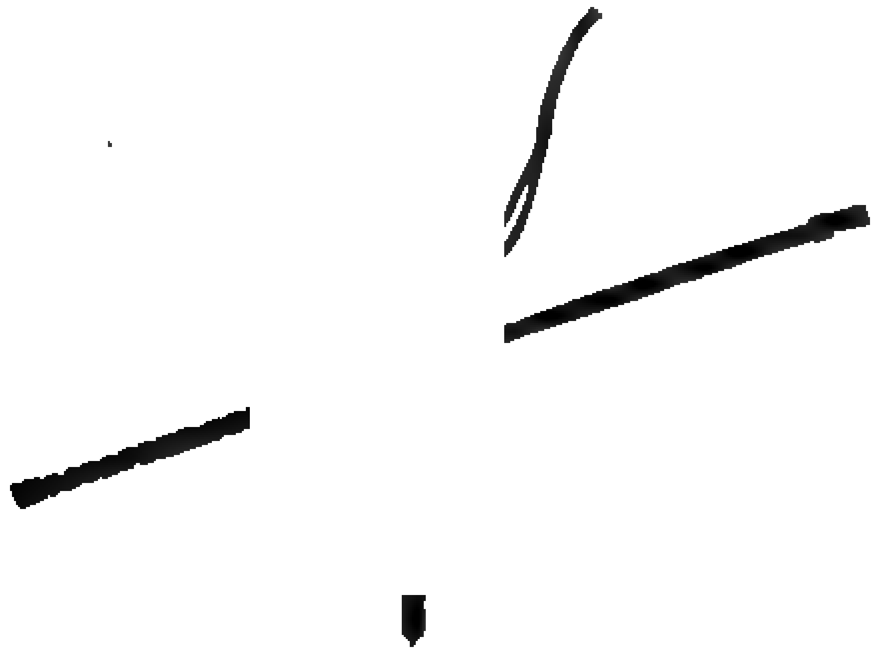


Fig. 295.—The Jeffrey Rotary Electric Coal-drill.

armature-shaft is a small pinion-gearing with a larger wheel. Two drills are used for each hole, one being 3 feet and the other 6 feet. The time required to drill a 6-foot hole in ordinary bituminous coal, with either type, is from twelve to fourteen minutes. In addition to gearing for hand-power drills there are ratchet attachments for driving them in such places where the holes must be drilled too close to reach the roof or the floor for the crank-motor to be used. It requires from ½ to 1 horse-power to drive them. The power is communicated through gears and a small pinion by which rotation is produced. A feather on the larger gear-wheel slides along a longitudinal slot in the feed-screw and permits of an advance of the bit with the progress of the boring. These force the drill forward at any determined rate without unduly crowding the bit.

Fig. 296.

A similar pattern of drill (Fig. 296) is operated by a small air-engine or by hand. In a hand-borer this drill is serviceable where electric or air coal-

cutters are also employed. Unless, however, the power-plants are to be installed for other purposes, it is hardly necessary to install one safely for the use of these drills, so long as equally serviceable hand-power drills can be obtained.

In loose material and shattered rock, shovels and spades are the only tools needed; clays and soft rocks require picks, crowbars, and shovels; ground that is scaly, brittle, and seamy is split by wedges in different ways. Massive rock cannot be broken without the hammer and drill. Powder and explosives are used in the latter case, and occasionally in the others, as auxiliary aid. For economy and speed, there are machines which are driven by steam, electric, or pneumatic power, imitating the operations of the above hand tools.

Underholing Bituminous Coal.—The operation called undercutting, underholing, bearing in, or kirving, consists in cutting a groove underneath a mass of coal in the soft floor, or in the coal itself, to a depth about equal to the thickness of the bed. The groove is 9 inches high at the face and about 2 inches at the back. The coal is propped by sprags while the miner is engaged in digging a deep groove. The coal then freed below may be broken off by the pressure of the roof overnight, or its fall may be furthered by a few drill-holes loaded with powder. The block may also be sheared by grooves along the sides of the rooms or stalls, cut vertically from roof to floor. Sometimes all three methods are practised to break the coal down. Shearing is more expensive than undercutting, hence is used only for shallow, firm coal or under a very strong roof. The cleats in the coal materially assist in its breaking. Anthracite is too firm to be treated in this way; it must be blasted off the solid.

Shooting Off the Solid.—This term is employed to designate the character of blasting, which is in practice in the anthracite-coal mines wherever there is but one face exposed toward which a blasting agent may operate. In anthracite mines this practice is universal, the miner exploding the holes so that each succeeding one will have a free face for blasting. In flat beds he makes centre cuts (Chapter VII) on the face by blasting wedged-shaped

masses of coal. In pitching-beds he blasts in the coal near the floor and undermines the bed. By systematic work a keg of 25 lbs. of powder can thus blow down from 30 to 50 cars of coal. This practice is warranted in bituminous mines when the coal is to be delivered to the coke-ovens, but not when it is to be marketed for domestic use.

Shovels.—The ordinary shovels are made of iron plate rolled under a welding heat with an edge of steel, and ears drawn out for the handle. A concavity to the blade imparts stiffness and carrying capacity to the tool. Such a scoop-shaped shovel is preferred by the anthracite miner. The end of the shovel is square or pointed, according to its uses. The pointed blade with the long handle is the most used. That of the "diamond label" is the most popular; the width of blade in inches designates the commercial number or size of the shovel.

Picks.—The pick is variously known as a pike, mandril, slitter, hack, and mattock, according to its shape and length. It is of iron with steel tips or all steel. It has an opening or an eye in the centre, the ends being pointed to a pyramidal shape or a chiselled edge. The points or tips are hardened to suit the nature of the work to be done. They are the wearing parts and should be replacable. On this account the all-steel pick is not serviceable. Each sharpening after use consumes an amount of metal, which, in time, reduces it to an inconvenient length. An iron pick 14 inches long with two steel tips wedged into the ends gives longer service, for the points when worn out may be replaced. Removable points are sometimes supplied with picks. "Pick steel" is a special steel in bars $1\frac{1}{4}'' \times \frac{5}{8}''$ or $1\frac{1}{2}'' \times \frac{3}{4}''$ which are cut into suitable lengths.

The picks are generally made at the mine by welding into the iron lengths of steel wedged and pointed. When finished they are about 22 inches long, though some are 29 inches long, and in the hands of the "box" cutters are doing remarkable "jadding" (cutting the top). The weight of picks varies between 2 and 9 lbs., with $3\frac{1}{2}$ or 4 lbs. as an average, the heavier weights being used for downward cutting.

The eye of the pick is oval. It is formed by gashing the red-hot bar in the middle, upsetting it, cutting it open by a drift, and hammering out stout cheeks with abundant metal at the sides and a bearing surface for the handle as long as possible. All of the strain of prying by the points and handle falls upon the eye, which therefore must be stout.

The shape and weight of the pick depends on the service it is to give. There is the straight or the curved pick, the anchor- or the poll-pick, each having favor for certain work. Indeed, in the same mine several forms are to be seen, perhaps as a matter of individual prejudice. The straight pick assists the reach, the curved pick enables a more effective blow to be struck. So, for overhand work, for underholing, and for getting into corners, a straight-head pick is used, but does not strike as well in downward work as the curved. The curve of the latter should be an arc of a radius equal to the combined lengths of the arm and handle.

Usually the bituminous-coal miner has a light pick of 2 or 3 lbs. weight for the clay and a heavier one for cutting bony coal, slate, etc. The light-weight picks have one square chiselled edge and a pointed one. The anthracite picks are heavier. The poll-pick used for prying loose coal from the face or rock from the roof is as heavy as can be conveniently handled.

The pick is single- or double-pointed. In the first variety the other end is forged into a hammer-head, known as a "poll," and is used for breaking fragments or driving a wedge. The tapered or chiselled end of the pike is used to pry into and slit the rock with the leverage obtained from the length of handle. For soft ground the pointed end is long and slender.

The handle or helve of hickory or ash is a straight-grained, firm, well-trimmed stick. It is trimmed to the shape of the eye and wedged tight by a pair of iron feathers. The handle should be at right angles to the pick. When so fastened, the arcs drawn by the tips, using the handle-ends as centres, will have equal radii. The length of American handles is about 28 inches; those of the English are 30 to 35 inches.

Breaking Dimension Stone.—Quarrying at the surface involves the procural of blocks of stone of definite shape and size suitable for dressing by the stone-mason to desired dimensions. The process must be such as to ensure with some degree of accuracy the shape desired without injury, and the rupturing agents must be less violent than powder. Wedges were therefore used, though they are rapidly becoming obsolete. The primitive wedges were of dry or wet wood. Now they are of steel or iron. They were forced or hammered into the planes of cleavage of the rock, splitting the latter along the given directions.

In Fig. 306 is illustrated a block being split by a number of plugs. The wedges are driven by a sledge in succession, beginning at either end and working toward the centre, each one receiving a few taps in turn. The holes are put alternately long and short, and of a depth 12 to 20 inches, according to the thickness of the stone.

The rock may also be broken, by forming a rift along the face, by a sledge. The stone is then pounded along a given line rapidly, until the continual concussion overcomes the tenacity of the stone, and starts a crevice along the line into which a wedge may be inserted and driven.

The Plug and Feather are more efficient than the simple wedge. The plug is a flattened wedge and the feathers have two wedges each, with one face flat and the other curved. They are introduced into a drilled hole with their points up. The plug is hammered between them and the design is such that the feathers are forced apart at the bottom of the hole with greater power than at the top.

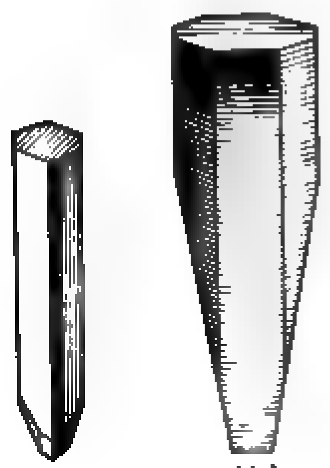
Hydraulic Wedges.—Hydraulic wedges have been used with good success at the collieries near Saarbruecken, the position of the driving-wedge being reversed from that of the plug, the thicker end being placed in the bottom of the hole and the edge near the top. Fitting between the two half-round wedged cheeks which point down, it is driven from below upward by the hydraulic pressure, or the force from the explosion of powder. This snaps the rock in the plane of the thin edge. Dimension stone is

obtained in some quarries by this process, but the risk of huffing off the rock at the top of the hole is great.

Gads or Moils are very useful accompaniments to every miner's equipment, differing from the early wedges only in that they have pointed tips, not chiselled edges. A piece of steel that has done service for drilling, has been dressed and smithed until it is less than 10 inches long, and can no longer be used for starting a hole, is converted into moil by tapering off the bit-point. It is used for chipping or trimming the rock, for a smooth bearing to timbers, etc. It is used in the Lake Superior copper-mines for "blockholing" or splitting the large masses of copper. It is almost indispensable in hard rock, and a number of them, sharpened and hardened, should always be on hand for the use of timbermen and shaftsmen.

A crowbar is a tool of occasional service underground. It is a long, unhardened, steel bar with a point at one end, and serves as a pry.

Jumpers and Drill-rods.—Percussion on the rock is accomplished by a jumper for vertical holes only or by a drill for holes



(a) (b) (c)

FIG. 297.—Forms of Bits.

of any angle. These are heavy steel bars provided with cutting edges, which may be straight, concave, or convex, and acute (slim) or "bluff" (Fig. 297), the shape being somewhat a matter of individual preference and skill. The bit is shaped by

the blacksmith, who hammers the red-hot end of a steel drill-rod to a wedge-shape, flaring it out to a width somewhat greater than that of the rod. The edge is worked to a straight or curved form, and then hardened by reheating the end to the proper degree and plunging it into a cooling fluid. The convex bit is stronger than the straight bit, and transmits the blow better with less risk of fracture at the corners. The cutting edge of ¾-inch drills is 1¼ inches wide, and other sizes are correspondingly flared. The other end of the drill-rod is not dressed after being cut off. It receives the blow of the striking hammer. When both ends of the drill-rod are worked with cutting edges the tool is known as a jumper.

The Jumper is a heavy 5- or 6-foot round iron bar, heavier near the middle and hammered to an edge at each end. A hole is churned down into the rock by lifting the jumper and allowing it to fall, after which it is turned, slightly raised, and dropped a foot or so. When one bit is dulled the jumper is reversed and the other end continues the operation. In this way a hole is "jumped down." The débris is cleaned out at intervals by a scraper or a spoon. Coal and limestone are churned in this way at a rate of 40 or 50 feet of hole in a shift; granite, at 15 feet. The jumper, with one cutter and a head for hammering, has lost the significance of its name and is properly a drill.

The Drill.—The steel rods for hand- or power-drills are procured in lengths of 14 feet and of any diameter. Though there are many grades, the English Jessop and the American brand "Black Diamond" have pronounced adherents. The steel is round or octagonal, the diameters usually being between ¾ and 2 inches. The length of the bar depends upon the depth of the hole to be drilled. The rods of less than 1 inch are used only for short holes and light hammering, as when a miner both holds the drill and strikes it with a hammer. In "double-hand work," where a companion strikes, the diameter of the drill is between 1 and 1½ inches. For "three-hand" or "double-hammer" work the steel may be as large as 2 inches, though it is doubtful if a hand-drilled hole greater than 1¼ inches

diameter is justifiable. Generally speaking it is preferable to have the drill as large as possible to transmit the blow, and also to produce a hole as broad as convenient. The drill of small diameter is rather light and yields under the blow without transmitting the effect.

Drilling Operations.—The hole is drilled by the repeated percussive blows upon the drill by the hammer. The rock is chipped by the concussion of the bit; after each stroke the drill is turned slightly and the blow repeated. Though the hole may be drilled without the addition of any water, it is nevertheless the practice to occasionally introduce some water into the hole to preserve the temper of the tool and to facilitate the removal of the débris.

The lengths of drills employed increase with the depth of the hole and its progress, the starting drill being a short, stout drill of comparatively large diameter, and the finishing drill, 30 to 36 inches in diameter, being of smaller diameter. The several lengths of drills constitute a set, the bits of a set being flared less with each increasing length of drill.

The depth of holes varies with the character of the work. It rarely exceeds 36 inches with double-hand work and averages 25 inches in single-hand work. In hard, brittle rock the holes are made deep and narrow, while in the tough and fissured material they are short and broad. Deep holes are more economical, but the miner usually refrains from drilling to the limit, except upon rare occasions, because of the difficulty arising from leaving the ground in poor shape for subsequent blasting, as will be seen in the Chapter on Blasting. The miner places the direction of the holes and carries them to such a depth as will enable him not only to produce a maximum effect, but also to leave a suitable free face for attack by the subsequent shots. As to diameter, it is cheaper to drill a small hole and to increase the strength of powder than to have a hole of large diameter with a large quantity of weak powder. The work of drilling a 1¼-inch hole is nearly three times that of putting down a ¾-inch hole of the same length; the relative volumes of

the rock pulverized are as the square of the diameters; or, in other words, all else being equal, 25 lineal feet of small holes can be put down while 9 feet of the 1¼-inch hole are being drilled.

Twelve feet, in the aggregate, constitute a stent with double-hammer work, and 9 feet of holes with the single hammer and one striker. In the case of single-hand work the bits are smaller in diameter than in the double-hand work for a gang of two strikers. About 30 inches of holes can be drilled per shift by single-hand work in rock of a medium character. In rare exceptions the progress may be as great as 5 feet per man.

The consumption of steel varies with the tool, but an average may be given as about 1 lb. for 15 tons of rock blasted.

Removing the Drillings.—During the progress of drilling the débris is removed by a squib, stick, spoon, or gun, according to the circumstances. The spoon is a round ⅜-inch iron bar, 40 inches long, having a handle at one end and a spoon at the other, which latter is made by curving slightly the lower 5 or 6 inches of its length and bending up a spoon at the end. The gun is a syringe, made of a length of gas-pipe and fitted with a suction-piston having a handle on its rod. By these appliances the load may be removed from the hole more effectually than can be done by the spoon. The latter is employed for scraping the sand from a dry hole. The swab-stick is sometimes used to remove the last remnant of moisture in the hole prior to charging it with powder.

Hammers are employed for breaking large fragments of rock and for striking drills. In weight and finish of their faces they differ. The face is usually flat and very hard. The convex striking-face is rarely used. Striking-hammers are not used for rock-breaking, which duty is left to the sledges.

For single-hand work the hammer-head weighs 3½ to 5 lbs.; for double-hand work, 5 to 7 lbs., except for wedge-driving; it is short with a broad face, with a handle correspondingly short or long.

Single-hand and Double-hand Work.—By single-hand work is understood the method of drilling holes by a single individual,

who holds the drill in one hand and strikes it with a hammer in the other. In double-hand work the operations of holding and turning the drill and of striking the tool are performed by two men. In double-hammer work the gang is composed of three men—two striking and one holding. Except in very hard rock, in which satisfactory progress is made by double-hammer work, single-hand work is the usual practice in narrow places and double-hand work where the room is sufficient to allow of swinging the sledge for a blow. For many reasons the double-hand work is almost universal. Though many objections obtain to its practice, it has an advantage in enabling the men to alternate the work of striking and holding and thus relieve one another. But the benefits derived from the single-hand work are so great that where it is possible of introduction it is employed.

Drinker says that in the point of economy of time and money "one-hand drilling" is from 30 per cent in soft schist to 20 per cent in soft sandstone cheaper than "two-hand drilling." In hard rock "one-hand drilling" gives the more rapid advance. Dr. A. Serlow, in his "Leitfaden zur Bergbaulunde," believes that except in shaftwork all other forms of drilling may be executed more accurately by the "single" than by the "double-hand," and perhaps more cheaply.

Itemizing the Drilling Expenses.—There are various methods of apportioning the cost of broken ground. One simple method consists in keeping the supplies, such as drill steel, tools, etc., under one heading; the power, fuse, caps, oil, and candles as sundries; the labor by itself; while the fixed charges, such as office and shipment expenditures, are given separate accounts. Occasionally the labor item is subdivided and distributed as a guide for future economies. It may be possible thereby also to ascertain the number of tons or cars broken per miner, from which may be fixed a standard of settlement. Again, the cost-sheet may be divided into accounts as follows: Under labor are miners, drilling, blasting, trammers, blacksmiths, engineers, timbermen, trackmen, engineering, and superintendence. The minerals-account contains items for powder, caps, and fuse, oils, waste,

candle, coal, stable repairs, track tools, and timber; and the miscellaneous account includes office expenses, taxes, insurance, sinking-fund, interest, and shipments. These different items are reduced to cost per ton of product as well as the aggregate cost per week or month.

The Blacksmith Shop.—The duties and work of the blacksmith may not seem relevant to the engineer, but no manager can afford to be ignorant of any element connected with the economy of his work. As a matter of fact, it is highly essential that the latter be capable of judging the performance of the blacksmith, who is the butt between the complaints of a miner's inefficiency and that man's retorts in pleading bad tools. A smith who can sharpen tools suitably for hard ground is held in high esteem by miners. The equipment of the shop comprises a full kit of tools, costing perhaps $30.00, a good bellows and tuyeres, Peter's anvil, vise-taps and dies, twist-drills and hoop iron, an assortment of carriage- and machine-bolts, screws, spikes, nails, a few horseshoeing tools, benches, etc. The building is about $14' \times 12'$, with hinged door-openings, near or over the fire and in the two walls, for working long bars. If machine-drills are used in the mine, a set of special swages are necessary for finishing the bits to the X, Z, or + form, as required.

The miners, who usually must pay for the wasted steel or for the sharpening of bits, are entitled to the services of a competent blacksmith and suitable drill steel. The charge is usually $1 per month in coal-mines and 3 cents per bit in metalliferous mines. The sharpening is often done, even for contractors, at the expense of the mine.

For a small mine employing 20 men in all, 1 blacksmith will suffice, though, of course, it depends upon what he must do. A good sharpener can dress tools for 20 men on medium rock, or swage the I or X bits for 7 machine-drills. Excepting the pointing of picks, the cutting of steel, and the handling of large pieces, he will need no striker. With this help he can make 12 heavy picks, 20 light ones, or weld 40 pick-stems in a shift; or he can finish 2 sets of colliers' tools of 5 coal-picks,

2 wedges, a hammer, and 2 bottom-picks. Alone, he can dress 40 bits an hour; with help, he can forge 25 double hand-bits, or draw out and temper 50 pick-points per hour.

Forge Fuel.—One important element of success in forging iron is a clean, pure fuel. This may be a slightly caking coal that gives flame and a high heat, or coke which is hotter but more difficult to keep alight. The coal should be clear of shale and slate, for they fuse and make a pasty cinder that adheres to the iron. It must be free of sulphur, which makes the iron "hot short" and also tends to produce scales while forging. For the latter reason white-ash coal is preferred to red-ash coal.

Welding.—A very useful property of wrought iron is its capability of welding, by which two short lengths may be united to form a bar of a serviceable length. The process consists of wedge-tapering an end of each bar, heating them to red, and subsequently hammering the softened parts together. A more difficult joint, known as the split, is described further on in the steeling of picks. If the welding has been well done, the point of union is as strong as any other part of the bar. Precaution must be taken to keep the surfaces clean and free from scales, which would interfere with perfect welding contact and are so apt to form in a thin fire of the forge. Scales are due to the oxidation of the iron, which while red-hot is not sufficiently surrounded by ignited carbon to consume the free oxygen of the air. When the layer of fuel is thin, or where too much blast is given, the nascent iron absorbs the oxygen. Once formed, the scales cannot be melted or fused off, but their formation may be prevented by a liberal covering of fuel over the iron or by sprinkling borax over the surfaces during the heat. Sometimes sand may do instead of borax, though it requires a higher temperature for its fusion. With proper regulation of fire, steel may also be welded to iron with advantage; as, for example, the steel ends of a pick, which receive the wear of the rock, are welded into the iron head of the pick-eye.

Steel is a compound of carbon with iron in varying proportions. H. M. Howe, in his "Metallurgy of Steel," says that steel,

in its specific sense, is "a compound of iron possessing or capable of possessing decided hardness simultaneously with a valuable degree of toughness when hot or when cold, or both. It includes, primarily, compounds of iron combined with from, say, 0.3 to 2 per cent carbon, which can be rendered decidedly soft and tough or intensely hard by slow and rapid cooling, respectively; and, secondarily, compounds of iron with chromium, tungsten, manganese, titanium, and other elementary compounds, which, like carbon steel, possess intense hardness with decided toughness." "The terms 'iron' and 'steel' are employed so ambiguously and inconsiderately that it is to-day impossible to arrange all varieties under a simple classification." The various adjectives qualifying the term "spring," "shear," etc., apply to the uses to which the steel is put, and imply a certain percentage of carbon constituency.

Hardening Steel.—The homogeneity of steel and the presence of carbon imparts to it a capability of hardening and tempering to a degree depending on the temperatures of the heating and the subsequent cooling. As the amount of carbon increases, the melting-point of the iron decreases; and this greater fusibility reduces its welding quality.

A steel is said to be "hardened" if, when red-hot, it is suddenly cooled. The reason for this change is not readily understood, though it is in some degree owing to the presence of the carbon; for pure malleable iron is not in the least affected by the operation, while both steel and cast iron are hardened to a marked degree. That attained by cast iron by plunging it into a cooling fluid is not so great as with steel. The greater the difference in temperature between the steel and the cooling fluid and the shorter the time of hardening, the harder it becomes.

The fluids used for the cooling bath are water, oil, mercury, and lead, according to the care to be exercised in the process. Water, having the highest specific heat, performs the operation of cooling most quickly. The steel, however, scales somewhat and becomes excessively brittle. The oil is of slower action and is believed also to supply from its decomposition some carbon,

which carburizes with the steel. Lead and mercury are used in large establishments.

Tempering.—This process follows hardening. The use to which the tool is to be put determines the degree of hardness desired, but the latter cannot be secured accurately without skill in one operation when water is used. It may be attained by a bath of molten lead for certain uses, but usually the steel is subjected to a second operation of annealing.

The hardened steel is again heated to a red heat and thus is softened slightly. Its point is then plunged for an instant into a cooling fluid to harden the lower end. Withdrawing the article and rubbing off the scales, the heat from above the end will be conducted to the edge. A series of colors plays over the surface in succession as the temperature increases. Beginning with a light straw, passing through the shades of yellow, brown, purple, and blue, the effects of the chill are partially removed, till, if the end should become red, all its hardness will have been removed. When the steel point assumes the color desired, leaving a given degree of hardness, the tool is finally cooled by immersion. A tool plunged at a straw is very hard, while one allowed to anneal by reheating to a blue is quite soft.

Caution is urged that the plunged tool while tempering be not held too long a time at a certain color-line, for it has a strong tendency to break there when in use. The tool should be slightly waved in the water. Pieces which are to be tempered throughout must be allowed to "soak"; i.e., become uniformly hot before plunging.

The Correct Temper for Tools.—The proper color for a given ground is only ascertained by experience. Generally speaking, the picks and drills are stopped at a straw if intended for hard rock and carried nearly to a blue for mild ground. It is always desirable to preserve the toughness of the steel as far as possible; therefore the lowest color is selected which is compatible with the service to be performed. A high-carbon steel is given a lighter color than steel of low carbon.

Metal-working tools are given a pale straw-yellow; wood-

working tools, a brownish tint; hatchets, saws, etc., a light purple; picks, to a rose; cold-chisels, to an orange-rose; keydrifts, orange; rock-drills, yellow-orange; screw-cutting dies, light yellow, and hammer faces, a pale straw. Few, if any, miners' tools are carried to a blue temper. They would be too soft. Crowbars should never be hardened, as they would be too brittle under severe strain.

Bits which have been properly tempered will wear down uniformly till too blunt for further service. Those which are too brittle will be found to have broken or cracked off. The former might be again tempered at a lighter color and the latter should have been softened further during the sharpening.

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CHAPTER VII.

CHANNELERS, DRILLS, AND COAL-CUTTERS.

Machine Rock-cutters.—The successful substitution of machinery for hand labor has proven a most important advance in mining engineering. Hard rock is no longer an obstacle to extraction of fuel, ore, or rock, and very long and large tunnels are now rendered possible in a comparatively short time. The opening of mines is accomplished in such short time and with more prompt returns for the investment that machine drills are employed to the exclusion of hand labor for driving shafts and tunnels. Every form of hand-labor tool has been successfully imitated and extensively introduced. The quarry methods of lewising the jumper, saw, chisel, pick, and auger find their counterparts in the channeler, percussive drill, coal-cutter, and diamond-drill.

Quarrying.—The quarrying of dimension stone was formerly accomplished by the trenching along lines decided upon. Carried often to a 10-foot depth and wide enough for a man to operate his pick, these trenches wasted much good material. These trenches are now replaced by channelers and gadders, which dig as deep as desired, but only 2 or 3 inches wide. These machines are mounted in different styles and cut perfectly true lines at any angle with or across the strata.

Channelers.—For extensive quarries these machines are mounted on a portable slining carriage, with boiler, rails, etc., and a feed which automatically moves it with the progress of its channel. A set (gang) of five cutters receives a reciprocating

FIG. 298.—A Single-gang Channeler.

motion from a steam-piston, through a connecting-rod, or through some yielding contrivance from the cross-head of the engine. The latter gives an elastic blow to the cutters. Automatic contrivances keep the cutters to their work. Machines are also supplied for cutting two channels at a desired distance apart; these are known as "double-gang machines" and cost

Fig. 299.—A Broach-bit.

from $1200 to $2000 complete. With three men and 400 lbs. of coal, at 150 strokes per minute, they cut from 75 to 400 sq. ft. of stone—the former in marble, the latter in soft lime—and replace fifty men (Fig. 298).

The Broach-bit.—Many quarries employ a small machine similar to the channeler mounted on a transversing and long stout bar which maintains the alignment of the work. Its frame (Fig. 299) is comparatively light and is adjustable to a high or low position. Its bit may be a broad channeler bit and drive rectangular holes in any direction and to any desired depth, or, by permitting the rotation of the bit with each blow, the ordinary

round hole may be drilled by the same machine. The broach-bit proper is a channeler-bit which breaks down the partitions between the holes drilled by a previous bit, as in Fig. 299. By this machine the channeler is cut to any length and depth desired, the line between the holes being directed by the stout bar of the machine. This machine is also employed for drilling holes for the plugger and feather work. By it 300 linear feet of 2-foot holes are put in ten hours, or 70 square feet of channeling in sandstone and 28 square feet in granite.

Lewising is accomplished by machine by fitting a tripod with a slot movement for the drill body such that three parallel holes can be drilled with a single setting.

Power Drills.—Reciprocating drills employing steam or air as a motor fluid are of common use for piercing the rock. The construction of the drill is the same for either fluid, though the name given to the drill varies with the fluid used. A cylinder of cast iron is capable of sliding in a guide bed-plate (Fig. 300) which is mounted on a tripod or a column. The cutting-tool is clamped to an extension of the piston-rod and the balance of the mechanism is such as is necessary to obtain a return of the drill with its progress and reversal of the motion of the piston. The essentials for a successful drill are simplicity in construction; as few moving parts as possible; a strength commensurate with light weight and easy transportation; variable stroke to the piston, whose blow shall be cushioned; and an automatic return and revolution of the drill. The weight of the machine must be sufficient to receive securely a shock of the blow. Thus a piston having a diameter of 2½ inches operating under an air pressure of 60 lbs. per square inch would receive a constant accelerating force of 294 lbs. with every stroke. To counteract this recoil its mass must exceed this in amount. As the steam- or air-drill has its largest sphere of usefulness in remote mining districts it should be durable and require a minimum of repairs.

The motor fluid underground is usually compressed air and on the surface steam, with an ordinary pressure of 50 to 80 lbs.

per square inch. The horse-power of the drill is estimated as a simple steam-engine, with the important difference that the ratio of the area of piston-rod to piston is larger. Again, the steam-

FIG. 300.—A Steam- or Air-drill.

engine does its work throughout the entire stroke, but the drill-engine only at the end of its stroke. Hence it can never work expansively. The air enters the cylinder and propels the piston

to the end of its stroke, when the attached drill strikes the rock. At that moment the piston automatically reverses its own valve, which admits air at the lower end of the cylinder, while a ratchet and spiral device slightly turn the tool, which is being drawn back for the next blow. As the work to be done on the return-stroke is merely to lift the tool, the annular area of the piston is much less than that on the other side, and little power is consumed. At the proper point in the up-stroke the valves are again reversed and the operation repeated.

The Automatic Steam-valve.—The main difference between the drills upon the market is in construction of the valve and the means by which it gives a reversal of motion to the piston. It is desirable for rapid penetration that the number of blows per minute should be a maximum, but structural difficulties place a limit to the length of time between each blow. It is essential that the full air pressure be exerted at the instant of the impact and that the valve should not reverse until the blow has been struck. Then it should reverse instantly, permitting only sufficient steam to remain in the clearance space under the piston to safely cushion the blow against the front head of the cylinder. The average speed is about 200 blows per minute. The frequency of the blow varies with the ability of the machine and is altered to suit the hardness of the rock. A short stroke, light blow, and rapid rate give the best progress in hard rock, and a hard blow is best in soft rock, provided the drill does not "stick" in the hole.

There are two systems of moving the valves. The first requires reversing rod and tappets, while the second is a duplex system requiring a fluid to reverse the motion.

The Tappet-valve.—In the earlier forms of drills the valve was operated by means of an external rod with an exposed three-armed tappet moved by a projection on the piston-rod, as in the steam-pump. The rate of speed of a drill, however, was too high for the mechanism to withstand the numerous and violent shocks ensuing at the time of the reversal of the valve.

The later forms of the tappet are concealed. The arc of its motion is reduced and a more compact machine is the result (Fig. 301).

The valve motion was positive, and though there are many disadvantages inseparately connected with it, it is safer in the hands of unskilled labor. This accounts for its retention in our present successful forms.

FIG. 301.—An Internal Tappet-valve Air-drill.

The little Rand giant drill (Fig. 300) is a tappet-moved valve and requires less dead space and consumes less steam than the Burleigh pattern, which it resembles, whose piston operates two rockers to oscillate its valve. Centrally in the cylinder a three-toed rocker is located, the upper toe being fitted into a recess in the valve which it moves and the two lower toes being alternately rocked by the piston, to produce the reversal of motion. By thus separating the spindles from the valves and the tappets which they connect, a greater durability was obtained than in the earlier types of drills. In the Sergeant tappet-drill the valve and the rocker are also in a single three-toed piece, driven alternately by either end of the piston.

In the Sergeant (Fig. 303) the piston-valve is moved by exhaust steam from the opposite ends. An auxiliary slide-valve moves over the arc of a circle by shoulders on the

piston, opens and closes the ports, and is a trigger regulating the movement of the main valve. There are no openings in the side of the cylinder and no ports for the piston to close; the exhaust remains open at one end till the blow is struck, when the valve reverses immediately.

The Fluid-driven Valve.—This is a piston-valve which reverses the motion of the piston by opening the appropriate ports for admission and exhaust of steam or compressed air, being itself cause to oscillate in its valve-chest by the motor fluid which is automatically admitted to one head or the other of its piston. It is adopted in the Rand "Slugger" (Fig. 302), the Ingersoll "Eclipse" (Fig. 303), the Schram drill, and others. The first two named are of similar construction and action, and are shown in section. Two port-holes connect the annular groove in the piston with each opposite end of the valve-chest and are opened or closed by the piston passing over them; the supply for one end and the exhaust to the other end of the valve-chest are simultaneously opened. The annular groove, therefore, is a general exhaust outlet for the valve steam, while the motor steam is exhausted by the valve connecting the inlet passage with the exhaust-pipe.

There are no means on the piston in Fig. 302 for maintaining a steam-tight or air-tight separation of the two ends. When wear occurs the steam pressure is lost and leakage ensues; the exhaust becomes imperfect and the valves may fail to act properly. When in practice the machine is found not to reciprocate properly the fault is usually found to be due to wear.

The Schram Drill.—An account of the Schram and the Darlington drills is to be found in André's "Mining Machinery," from which the following is taken: "Schram's consists of a slide-valve and a slide-rod that admit steam to the cylinder for raising the piston and drill. When the piston passes a certain front porthole, steam enters through it into the back of the valve-chest at the same time that the front valve-chest, through the other port and the hollow circular groove of the piston, communicates with the exhaust-pipe. Steam then works full pressure on the

FIGS. 302 and 303.—Fluid-moved Valve Air-drills.

slide cylindrical rod, which, with the slide-valve, is forced towards the front valve-chest, so that the back steam passage is open to the cylinder, and the front steam passage connects with the exhaust-pipe. The piston moves forward, and, when it passes the back port, allows the steam to enter the front valve-chest at the same time that the back valve-chest, through its back port and the circular groove of the piston, communicates with the exhaust. The slide-rod is forced back, the front steam passage opens, and the back passage communicates with the exhaust. The slide is in the form of two spindle-valves, so that it remains in position without recoil, and the annular groove of the piston is always in communication with the exhaust.

"**The Darlington Drill** has only two working parts,—an extreme of simplicity: a cylinder and its cover, and a piston and its rod. The piston is made to operate as a valve. The inlet-pipe, having open connection with the cylinder, *always* furnishes the pressure to lift the drill, which rises whenever there is no pressure on the back. On its way up, the piston first covers the exhaust (above the inlet), and then uncovers an equilibrium passage, by means of which communication is established between the front and back ends of the cylinder. Then air or steam enters and operates over the greater area, at the back, and first checks the upward movement, soon overcomes it, and finally produces a forward motion. The propelling force, now, is dependent upon the difference of area between the back and front of the piston. On its way down it soon cuts off the equilibrium passage and the air can only enter at the inlet; the steam operates by expansion for a short space, till the piston has passed and uncovered the exhaust-port, when a discharge takes place as the blow is being struck. One fact is noticeable, that the amount of steam used is only that necessary for the down stroke; for that used to raise the drill escapes by the equilibrium passage to the top."

The Drill-rods. — The steel drill tool is of a diameter depending on the size of piston, and from $\frac{3}{4}$ to $1\frac{1}{2}$ inches, according to the intensity of the blow to be struck. The smallest size is attached to a 2-inch piston and the largest size mentioned

to a 5-inch piston, corresponding to blows of 200 lbs. and 1200 lbs. respectively. The average size of the mining drill is a 3-inch piston with 1- or $1\frac{1}{4}$-inch steel. The drill-rod is clamped by a heavy split chuck locking into the enlarged end of the piston-rod, and reciprocates with it. The end of the drill or shank varies in dimensions according to the piston size. For a 2-inch-diameter piston the shank is $\frac{5}{8}'' \times \frac{3}{8}''$, for a 2-inch piston $1\frac{1}{2}'' \times 7''$, for a piston of $4\frac{1}{2}$-inch diameter and proportionate sizes for the intermediate diameters. It reciprocates through a split or solid spool front head, which is provided with a stuffing-box and a gland to prevent the escape of steam. The latter head is adapted to air-power, while the split fronts are preferred for steam-drills.

The bit is usually forged with a bluff cutter edge for the strength and is provided with the usual flare. For rocks which do not crush, but chip in large fragments, a sharper edge will do better execution. The cutter is dressed by special tools to the X, I, Z, or S form, each having its specific value, the S form being more likely to maintain a round hole.

A set of tools accompanying each drill is graded according to the amount of feed which the drill is capable of. The shortest and the stoutest drill of the set is about 15 inches in length and the other lengths are successively longer by the amount of feed of the machine. The longest drill in the set corresponds to the maximum depth of the hole. A machine feeding 15 inches and drilling a hole 10 feet deep will have 8 drills in a set, the difference between their lengths being 15 inches. For a 5-foot hole, 3 or 4 drills constitute the set. The bits should be hardened in such a manner as to have durability corresponding to the length of the feed. A bit usually will require sharpening after 4 inches of progress.

During drilling the tool must be turned uniformly, as otherwise the hole may become rifled. By rifling is understood the tendency of a drill to cut a triangular hole instead of circular hole, which it strikes successively at one point or along a given line. With the straight-edge cutter the tendency to rifling is stronger than with any other form. In rock of an unhomogeneous

character rifling is common. With deep holes this may also result in a deflection of the hole from a straight line.

To prevent rifling and to produce rotation of the drill through a small arc during each stroke, the fluted bar-nut constituting the long thread ratchet is a usual appliance. In the Burleigh and the Darlington drills the device is a spiral feather on the piston-rod, recessed into a grooved piece in the cylinder-head. It is toothed and held by a detent, which permits it to turn on the forward stroke, but prevents turning during the up stroke of the engine. In the Ingersoll drill a grooved bar fitting into the back of the piston turns it on the back stroke and is itself allowed to rotate on the down stroke (Fig. 304).

FIG. 304.—The Rifle-bar.

In the Darlington drill the ratchet turns the piston and drill on the up stroke, and itself turns during the down stroke.

In the Schram pattern an auxiliary piston turns the drill, the former being driven by the fluid in a manner similar to that of the valve.

The Feed.—Whatever may be the pattern of the drill, the method of feeding the cylinder and its piston to the work is the same. It is always under control of the drill-runner and is capable of adjustment to the progress in the rock. The cylinder and its tool is allowed to move in a guideway which is rigidly mounted in the frame by the means of a square thread feed-screw which is turned by a crank at the rear end of the machine. The drill-runner usually stands at work with one hand on the feed-handle crank and the other on the steam-throttle valve.

Cushioning the Blow.—The maximum percussive effect is obtained when the blow is struck under the full head of steam or air. Such a dead blow is highly desirable; but on account of the shock to the machine, for the subsequent repairs this is unadvisable. A clearance space is provided in which is enclosed

a small amount of air to act as the cushion or elastic buffers are inserted to terminate the stroke. The former is less expensive than the latter.

The Drill Supports.—A rigid support is a rigid adjunct to the drill, and several types of mountings are provided, each having a special end in view, though a machine can be shifted from one style to another. In tunnels and shafts where the ranges of holes have approximately parallel directions, it is clamped directly upon a stout, hollow, cylindrical column (Fig. 305), for upon an arm projectingrom it.

Fig. 305.—The Column Support for Drills.

This admits of drilling several holes from one position of support. Jack-screws at one end clamp the column while claws at the other end bear against blocks resting upon the rock. The column can be had in various lengths. It weighs about 30 lbs. per foot. When placed horizontally, as in shafts (Fig. 305), it is known as a shaft-bar. When used in the vertical position it is spoken of as a column.

The tripod is the more common drill support, having a wider range of position than the column because of the universal joint

at its head (Fig. 306). Its stability depends upon its weight, and additional dead weights are, therefore, clamped on each leg as in Fig. 300.

FIG. 306.—Blocking out Dimension Stone.

The drill has no intricate mechanism and can be intrusted even to unskilled labor. Care should be taken in beginning operations that the drill strikes squarely upon the face of the rock, and with short, light blows. It can do more work with the consumption of less powder, steel and blacksmith work than can hand-drills. It can be set up in any place capable of accommodating a double-hand gang of men, and gives little trouble in placing into position. It cannot be adopted in small veins or where the mineral is friable or occurs in thin streaks.

The comparative merits of the several types of machines cannot be stated, for in one mining-camp the Rand drill, and in another the Ingersoll drill, are exclusively used. The Waring drill, the National, and the Burleigh, are employed for the same operations in the different mines of the same camp with apparently equal success. One range prefers the little Giant, but in

another mine it is discarded for Slugger pattern of the same company. In like manner a preference may be displayed for the Eclipse- or Sergeant-drill made by the same company. They are all highly commended, and their improvement in the inland community may be a matter of accident or of natural selection after periods of test.

The fluid-moved valve-drill style seems the favorite pattern for hard rock; but whether under all circumstances it is the best, one would not dare to aver. The nature of the rock, the proper air pressure, the rate of speed, and the proportion of rotary motion necessary for a maximum effect vary so largely that a comparison of the numerous published results cannot be made.

Progress and Cost by Machine-drills.—An average of nine neighboring mines in conglomerated rock shows machine-driving and stoping to be, respectively, 22 and 36 per cent cheaper than hand, and sinking 4 per cent dearer, with the progress respectively 60, 54, and 38 per cent more rapid. The gain in time during sinking more than compensates for the cost. In iron-mines the product of the machine labor costs about one fourth as much as that of hand labor. The consumption of fuel or air per drill may be calculated in the same manner as for the ordinary steam-engine. The cost of the average mining drill is about $325, and a complete plant with its drills and suitable compressor, etc., is $7000.

Systems of Machine-drilling. — With the advance made in the use of the power-drill, the systems of drilling and blasting rock have undergone corresponding changes. In hand-work the object sought when placing a hole for drilling and blasting is as much to secure a good bench for the succeeding shot as to break a maximum of ground. With machine-drilling, simultaneous shooting is practised and numerous holes are drilled over the entire area of shaft or drift, and fired in volleys without regard to the probable condition of the face after the blast. The removal and the resetting of the machine after each blast occupy so much time that a maximum number of holes is drilled with each set-up, and the time thus saved more than compensates

for the value of the powder wasted by not conforming to the fundamental principles of blasting, as in single shooting.

FIG. 307.—The Centre-cut System.

Systems of Drilling Holes.—There are two general systems in practice for the drilling of holes, the first being known as the centre-cut system, employed for tunnels and shafts of an area

larger than 7′×5′. The second system is known as the Brain radial system, which is employed in headings only large enough to accommodate one machine. In either system as many machines as possible are set up against the face. Three are employed in a single-track tunnel; two in a 11-foot heading; and as many as six are arranged in a double-track tunnel 27 feet wide. It is not unusual to mount two drills on a single column. The holes are drilled as determined upon, and after loading are blasted for an advance of 6 to 10 feet with each set-up.

The Centre-cut System is of almost universal acceptance in America. Vertical tiers of holes are drilled over the face, as in Fig. 307.

The method of placing the holes is shown in Fig. 308, the arrows indicating approximately the direction in which they are drilled. The top or back holes *aa* are pointed a little upward in order to obtain the top height and to break down the rock left after the other holes have been fired. The side holes *bb* are pointed a little upward and outward to obtain the full width of tunnel after the centre holes are fired. The centre tiers of holes *cc* are pointed centreward, the bottom ones of the tier being given an upward slant, the top ones a downward slant. Sometimes a plunger-hole *d* is drilled and fired with the centre holes. The holes are fired in rounds. Those at the centre, having the hardest work, are charged with a higher grade of powder and throw out a wedged-shaped mass; the side tiers *b*, being charged with a lower grade of dynamite, are fired later together, after which the back and trimming-holes are exploded together to complete the cut.

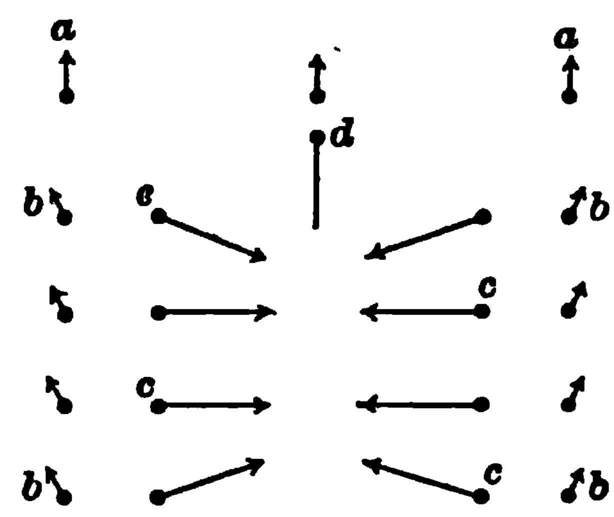

Fig. 308.—Direction of Holes for the Centre-cut System for Wide Gangways.

In the use of the centre-cut system for wide tunnels, in connection with the American methods of tunnelling, as illustrated in Figs. 309 and 264, the centre holes 11 (Fig. 310), fired first,

facilitate the work of the next two rows 2 2, after which the holes 3 3 are fired in volleys, succeeded by the trimming-holes 4 4.

The profile should be finished with the advance of the face, regularly checked up by the surveyors, for it seems difficult to maintain the proper floor level, the tendency invariably being to gradually raise the floor with the progress of the work in an unconscious effort to insure drainage.

FIG. 309.—Drilling Bench-holes in the American System of Tunnelling.

In driving such a tunnel the work is conducted in three shifts of eight hours each. During the first shift the drill-runners and their helpers are engaged in operating the machines and drilling the holes to their proper depths. The powder-men follow; they remove the machines from the face to a safe distance, and charge the holes with explosive and ignite them. In the third shift the broken material is removed by loaders, who tram the car sfrom the face to the point of discharge and also deliver the drill and appliances to the face for the drillers.

Drilling in Benches.—The bench of the tunnels may be attacked in two sections, as in Fig. 309, *A* and *B*, or in one only, as in Fig. 310. In the former, two wall-holes, one or two transverse rows of four top holes downward, and half a dozen bottom holes, lift each bench with every other shift. Fifty-four feet of progress a week is the record on a very hard sandstone bench 14′×27′. This bench-work is accomplished, not only more

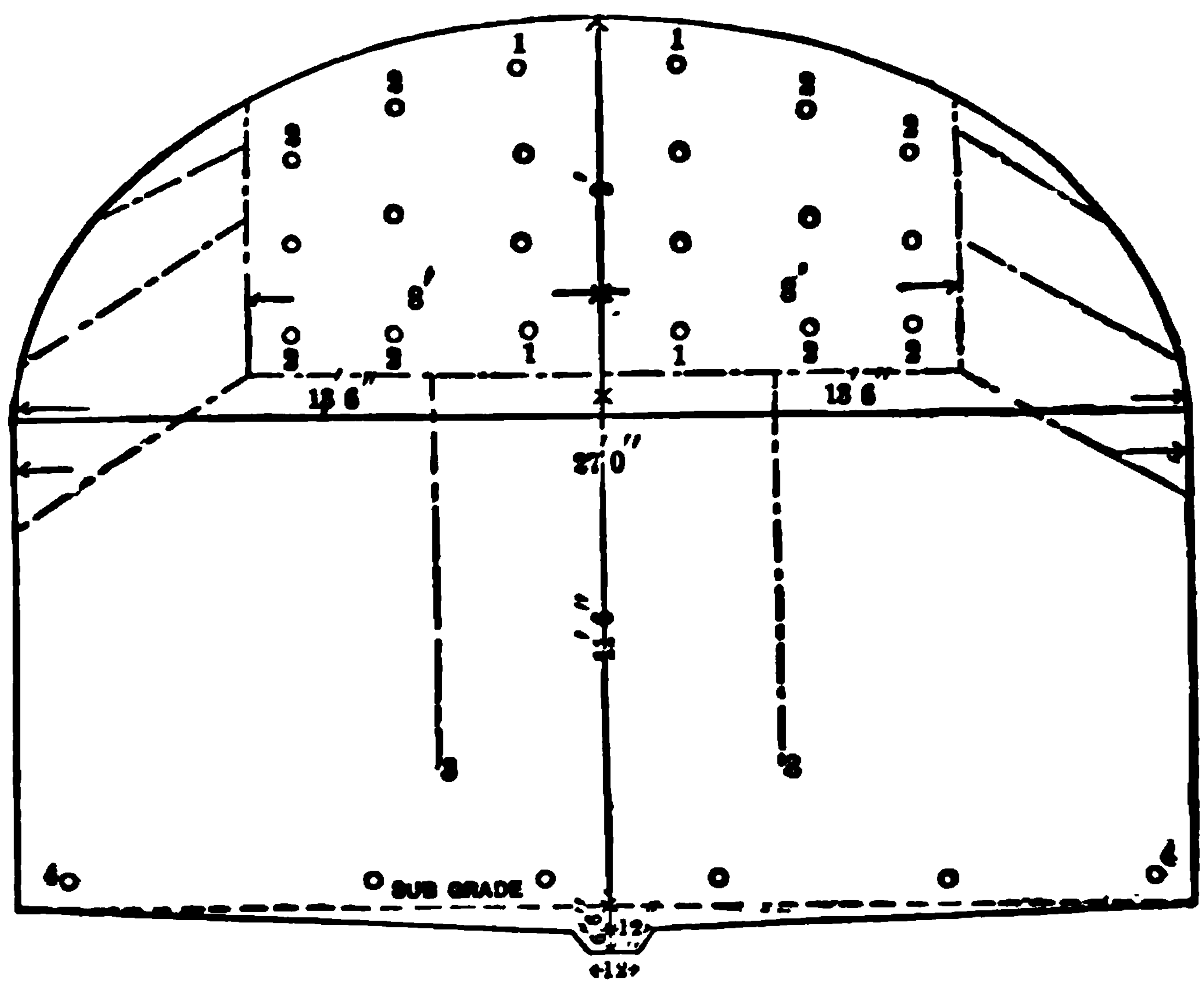

FIG. 310.—The Centre-cut System of Drilling Wide Tunnels.

rapidly, but also with a powder consumption per cubic yard of rock of about one half that in the heading above it.

The Centre-cut System in Shafts.—The method of placing holes in shafts does not materially differ from that in the headings. In Fig. 311 is the elevation of a 6-foot cut in a shaft whose area is 10′×20′. The centre holes are placed 10 feet apart approaching, but not intersecting. Six pairs of holes, spaced 2 feet apart, extend across under the shaft. The holes *bb* are in two pairs of rows of five each, the first being given a greater inclination than the second row. They are fired in separate tiers of ten holes each. Occasionally all of the twenty holes are fired

together, followed by the squaring-up holes *cc*. For a deeper cut the centre holes (Fig. 312) are in two tiers, those in tier *a* nearly meeting at 6 feet from the shaft floor and the other *bb* at 11 feet. The other holes are driven as before. In this case the 6-foot centre is opened first, followed in sections to the sides by the others.

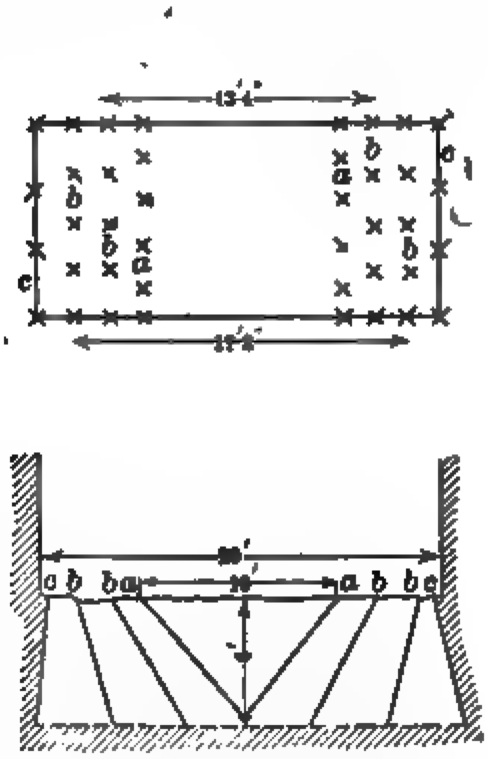

FIG. 311.—The Six-foot Centre Cut for Shafts.

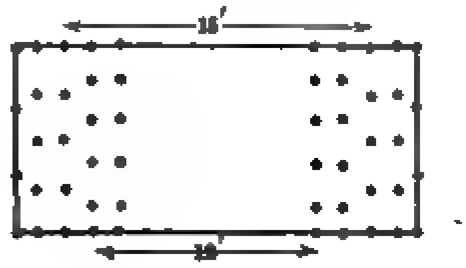

FIG. 312.—The Ten-foot Centre Cut for Shaft-sinking.

Brain's Radial System.—This is designed for small faces which can accommodate but one machine at a time. Like the centre-cut system, it is equally applicable to shafts. All the holes to be drilled are put from one position of the machine, which is removed only when firing is to be done. This minimizes the time lost in shifting. In a certain case the machine, from a position 4 feet 8 inches from the bottom, 2 feet from the top, and 2 feet 6 inches back from the face, put twenty-nine holes with a total length 70 feet, advancing 3 feet with an average of 2.4 cubic feet broken rock per lineal foot of hole.

The holes are comparatively shallow, though they vary greatly in length, those making the smallest angle with the face being the longest. Four ranges of holes are drilled. Sometimes a few extra squaring-up and lifting holes are necessary to trim the

periphery of the drift, but ordinarily the firing of the most angling holes first breaks out the rock to daylight and opens a face for the other successive rounds. The advance cannot be large, for neither deep nor angling holes are possible in a narrow drift. In a drift 8 feet wide, two settings of the machine are sometimes made drilling from near each wall and thus forming a modified centre-cut plan. In some mines a practice prevails of cutting a horizontal range of bottom holes, two ranges of holes looking downward, and a top row to break out horizontal instead of vertical wedges; this plan requires a bar-mounting for the drill, in a drift of say $7' \times 8'$.

The Continuous Process of Diamond-drilling.—Gen. Henry Pleasant's method of shaft-sinking is a novel and eminently successful application of the diamond-drill. One or more diamond drilling-machines are set up over the site of the shaft, and bore vertical holes as deep as the shaft is to be carried. The machines are moved to new positions and additional long holes bored. The operation is continued until the entire area of the shaft is pierced by holes at suitable distances apart. One shaft had thirty-five holes bored over its area of $25' \, 8'' \times 13' \, 10''$ to a depth of 300 feet in six weeks by three machines.

When the "continuous process" is completed, the machines are removed for the blasting. The holes are filled with sand or water for the full length, except in the upper 3 or 4 feet, which are treated like short holes, charged with dualin and fired,—the central ones first. When the débris has been cleared away, the shaft will have advanced 3 or 4 feet. A few feet of each hole are again cleaned out, loaded, and fired. Thus each section advances with an uninterrupted alternation of shooting and hoisting. Though it is not always cheaper per cubic foot, it effects a great saving in time, and quick access under ground may prove the element essential to the success of the undertaking.

Boring Headings.—The diamond-drill cannot be used advantageously for tunnel work or driving headings. Percussion-drills are there used almost exclusively, though rotary machines have been employed for boring out a heading almost to full size. The

Stanley header represents this class of machine. At one operation a series of cutters on a rotating boring-head grinds away the whole face for the core of a heading some 7 feet in diameter. One was used in the Mersey subaqueous tunnel. It travelled at the rate of 39 inches per hour and executed its work satisfactorily in the argillaceous chalk.

The approximate cost of sinking and tunnelling work varies with the character of the rock, the depth, and excavation, of distance from the surface; the efficiency of the workmen also determines largely the economy. In the New house tunnel, with an area of 144 sq. ft., the cost of driving was $28.80 per foot, in the Wooster tunnel it was about $40 per foot, and in the Aspen tunnel $19 per foot. The cost of driving is usually greater through soft ground than through hard ground, on account of the increased quantity of timbers required and the difficulties of draining the water. The cost of driving a rock tunnel 5′ × 7′ on contract without timbering is $8 per foot. An upraise can be driven at a cost of between $5 and $10 per foot, varying with the size and the character of the timbering needed. A two-compartment shaft with the timbers can be sunk 6′ × 12′ in the clear at an expense of $25 per foot.

Electric Rock-drills.—The various attempts to introduce an electric percussion-drill to replaced compressed air-drills have resulted in only a partial success. The advantages which electricity present in transmission over compressed air are sufficient to suggest an installation of drills. The difficulty, however, lies in the inability to produce a reciprocating motion without the use of solenoid.

The Solenoid Drills.—Two coils of wire are wound in the form of a spiral through which a current is alternately passed in the one direction and the reverse. If a steel core is passed through the adjacent coils and a current of electricity be passed through one of them the coil will be repelled through the other coil. The latter being next charged, will give a reverse action to the coil and repel it toward the first. This sets up an oscillatory movement which, if the machine be perfect enough and the coils carry

sufficient current, will accomplish work similar to the percussive machine (Fig. 313).

FIG. 313.—A Solenoid Drill. FIG. 314.—A Gardner Drill.

The Siemens, Halske, and Gardner Electric Drills are based upon the attempt to produce indirectly a reciprocating motion. The current does not enter the drills, but drives gearing housed in the drill cylinder by means of a motor and a flexible shaft. The drill is mounted on an adjustable column arm or on a tripod (Fig. 314). This machine produces a rapid succession of light blows rather than a small number of heavy blows. It requires 2 to 3 horse-power to operate it. It is also quite heavy.

The Box Electric Drill dispenses with the flexible shaft. In the rear of the drill is a motor, geared by a pinion to a wheel on the crank-shaft. At the other end of the crank is a pinion which operates a train of gearing, turning the drill after each blow. Between the crank-shaft and the small cylinder is a connecting-rod.

Reciprocating motion is imparted to the cylinder by the crank. The heavy piston with a thick piston-rod constitutes the hammer whose weight determines the blow.

Each end of the moving cylinder is filled with air at atmospheric pressure, which being alternately compressed and rarified cushions the blow. Two ports located somewhat centrally at the

side of the moving cylinder are provided for admitting air whenever the piston uncovers them. The weight of the drill with its motor is about 350 lbs., the latter alone being 100 lbs.

The Adams All-steel Drill is simple in mechanism, has few parts easily gotten at, and all wear taken up, since every bearing is adjustable. The drill is actuated by a loose rod running through the gear-case and motor, and this imparts motion through a pair of bevel gears and crank-shaft to the draw-bar and piston, the latter being cushioned with gangs of helical springs which serve to absorb the shock of the blow to protect the parts from injury and also to store up energy on the back stroke, which it can expend in forcing the steel into the rock. These springs prove extremely useful when reaming or for pulling the drill out of a fissured hole. The drill, striking from 575 to 600 blows per minute, will consume less power for its work than an air-drill. It consumes three horse-power at the generator to operate it.

It is claimed that in point of first cost the electric drill has the advantage over an air-drill, for a one-drill plant complete, operated electrically, costs one half of an air compressor and drill. The simple and smooth-running dynamo is less noisy than the complicated air-compressor as a source of power. The objection to the electric drill, that it provides no means for ventilating the workings as does an air-drill, is met by the equally valuable advantage that light can be obtained from the same wire that supplies the power. The comparative efficiencies of the rotary electric drills, reciprocating electric drill with flexible shaft, the solenoid reciprocating drill, and the air reciprocating drill are respectively as 1 : 1.66 : 4.5 : 10.

Coal-cutting Machines.—Coal is mined by machine in the same manner as by hand labor. Holes may be bored by machinery and the powder charges in them exploded, or the coal may be undercut or sheared by machine and broken down with or without the aid of powder. For boring holes in the coal various types of portable augers are employed, such as shown in the following chapter. For the undermining of coal there

are two types of machines, one depending on percussion and the other on abrasion, as produced by the chisel-edged teeth. Shearing-machines are of a similar type of construction to the undermining machines. The percussive machines are identical in appearance and construction to the air- or steam-drills employed in rock with the exception of the character of the bit. Of abrasive machines there are three classes using sharpened teeth attached around a bar which advances broadside on the coal face, on a link chain advancing normally against the face, and on the edge of a rotary wheel at the side of the machine advancing parallel to its face. The first two classes are breast machines and the one last named is a longwall machine. The motor power for any of these types may be electricity or air.

The same character of labor that the miner performs in removing and cutting a groove is imitated by the undermining machines in the floor under the coal or on the bottom layer of the coal itself. This groove is carried as far back from the face as possible and over the entire width of the room. The shearing-machine cuts a vertical groove along either side of the rib into the coal face and to the depth the machine can reach, beginning at the roof and extending to the floor. The use of either of these machines removes from the miner's toil the most laborious portion of his work as well as the most hazardous phase of his occupation. The normal effort of the digger exerted under the unfavorable conditions existing in the room is most wastefully applied and produces in addition an excessive amount of fine coal, which is lost. The machine produces more coal with less waste in a much shorter time. It reduces the labor of the miner and results in subdividing the work formerly imposed upon one man and increases the efficiency of each man by specializing each branch of the work. From four to eight loaders follow each machine and a total of twelve men, on the average, are employed in a mine for each machine in service. Its introduction results in an increase in the earning capacity of each miner besides improving the conditions under which his labor is applied.

The introduction of machines decreases the number of delays

of standing shots. It gives a steadier output and concentrates the operations of the mine because less territory must be kept open for the desired capacity. It requires, however, a more systematic development of the mine.

In Illinois the Legg machine is used in driving-rooms, and elsewhere the Harrison, Jeffrey, Lechner, and Sullivan machines. In Europe the machines in vogue are known as the Marshall and Frith.

The Requirements of the Machine.—It should be light and capable of being handled by two men and occupy small floor space or height, to admit of being moved around and between the roof-props. It should be capable of starting in a corner of a pillar or a loose end, and of cutting a groove, from wall to wall of the room, to any height, right handed or left handed.

The Percussive Machines.—The reciprocating-drill is largely employed for underholing the coal in the Mississippi Valley States. It is often known as the punch-drill, and has as its chief representative the Harrison machine (Fig. 315), which requires little explanation. It is a modification of the compressed-air drill, having for a valve-motor a single rotary device. It is mounted on low wheels on a platform inclined toward the face to enable it to move freely while making a cut; it is guided to its work by handles on either side and is prevented from recoiling by wooden blocks back of the wheels. In addition to the drill-runner, one helper is necessary to remove the chippings from the face of the work. An air pressure of about 60 lbs. to the square inch is used expansively, the point of cut-off being fixed by the runner according to the hardness of the coal. About 200 strokes are delivered per minute. Usually three bits are supplied with each machine in lengths of 2, 4, and 6 feet. A considerable slack coal is produced while digging a deep undercut of 5 or 6 feet, its height being 16 or 18 inches at the face and 3 inches at the rear. The lateral dimensions and shape of the groove are determined by the reach of the machine, but the entire face of a room is entirely undercut by a succession of such grooves, for which purpose the platform and machine are shifted sidewise.

Fig. 316.—The Jeffrey Bar Coal-cutter.

The average work of the machine is 80 lineal feet of face in ten hours, which is equivalent to from 50 to 100 tons of coal per hour, according to the height of the seam. The machine is compact and can be employed where the roof is weak and the timbers are close to the face of attack. Its weight is from 500 to 700 lbs. and its height rarely exceeds 17 inches.

The Sergeant Drill Company has an acceptable adaptation of its drill for coal-work which is largely used in the Southern States.

A variety of pick-machine is in use by which a 75-lb. pick is swung by a bell-crank lever at a weight of 70 blows per minute, cutting a hundred square feet of ground 2 inches high and 42 inches deep.

The Breast-machine.—In this type the coal-cutter with its motor is mounted on a frame which is caused to slide, inside of a substantial base frame, toward the face or breast of the coal, and under it to the limit of the slide. The cutter consists of a number of hardened steel bits on an edge of a wheel or over a rotating-bar in front of the machine; or it may be a series of bits inserted into an endless link-belt. The wheel or bar is forced forward, cutting a width of grooves 24 to 44 inches to a depth of about 6 feet. The groove is from 2½ to 5 inches high. The chain is drawn horizontally and uniformly around four wheels at the corners of the sliding-frame, and as its cutters pass in front of the machine the coal is cut to about the same width and depth as above mentioned. After each cut the motor and its cutter are withdrawn, the standard frame is moved along the face a distance equal to the width of its groove, and from the new position the next advance is made. The power employed may be either air or electricity, the former being preferred, unless the mine contains considerable gas.

In Fig. 316 is shown the Jeffrey bar-cutter. This is a bar fitted with numerous cutting-chisels, rotated by a link-chain shown at the side of the machine. The bar-cutters are kept to their work by a feed at the rear; their frame is shown advanced slightly from its starting position. A pair of jacks, one at the

front and another at the rear, braced by a screw against roof and coal, gives stability to the machine. Moreover, the jack at the front in a measure assists in holding up the coal from bearing excessively upon the cutter-rod. This machine is still serviceable in some coals, though its cutters have a tendency to climb up into the coal. This is a great annoyance to the operator. It also acts in its abrasion across the bedding-plane of the coal instead of parallel to it, as does the chain-machine, and in this respect is not so efficient.

The more modern type of breast-machine manufactured by the Jeffrey Machine Company is illustrated in Fig. 317. This

Fig. 317.—A Jeffrey Breast-machine.

chain-cutter is the logical outcome of the bar-cutter of the past. An endless chain carries a number of bits made of $\frac{3}{4}$-inch steel, having a chisel cutting edge $\frac{7}{8}$ inch wide. The chain, revolved by the motor and suitable gearing, is advanced automatically. The bits project from the chain sufficiently to make a cut $4\frac{1}{2}$ inches on either side of it (Fig. 318). The machine weighs about 2000 lbs., but is easily removed to a new position by releasing the jacks and skidding the frame, after which the jacks are reset. The voltage of the electric current is 500, and 35 amperes are consumed. Excellent results are obtained in gaseous coal, splint coal, or block coal.

Longwall Machines.—By inserting the chisel-bits into the circumference of a rotary disc which is operated at high speed by the electric motor, projecting it from the side of the machine,

FIG. 318.—The Sullivan Coal-cutter.

a longwall cutter is produced by which the machine may be caused to move parallel to the face of the coal, while the wheel or toothed disc undercuts the coal for a depth equal to a little over half its diameter.

The Winstanley machine consists of a rotary toothed disc, capable of being tucked away under the carriage or turned out against the face and revolved by two oscillating cylinders. With an air pressure of 30 lbs. per sq. in. and a machine weighing 1500 lbs., mounted on a carriage moving along the track, the progress of the cutter is 70 sq. ft. of underholing in an hour.

The longwall machine placed on the market by the Jeffrey Machine Company is represented by Fig. 319. This is a one-rail machine, balanced in such a manner that the cutter-wheel may pass over all irregularities in the floor. A series of

FIG. 319.—A Longwall Machine.

gears are arranged, winding a rope on a drum to move the machine along the rail at a rate fixed by the operator between 8 and 25 inches per minute. The cutter-wheels are from 3 to 6 feet in diameter, according to the thickness of the coal-bed. The bits inserted on the circumference of the wheels make a cut of about 4 inches wide. On the top of the cutter-wheel are gear-teeth engaging with the pinion and the engine-shaft, by which the rate of rotation of the wheel can be regulated to that which is most suitable for the coal. This machine is said to undercut an average of 600 lineal feet of coal-face per day.

Shearing Coal-cutters.—The shearing of coal by machine may be accomplished by a percussive machine, mounted on high

wheels or by a chain-cutter turned in position to a vertical plane. The reciprocating-machine, when shearing coal, is mounted on wheels of a diameter of 3 feet, or even more, enabling it to assume a wide range of position in making the vertical cut from a high angle pointing upward to a steep angle pointing downward.

One variety of the shearing-chain cutter is shown in Fig. 320. It is mounted on four posts, each supplied with a jack-screw, to secure it to the roof and floor with suitable clamps that permit of a rotary movement about the column and an adjustment to any height. The armature-shaft of the motor is parallel to the centre rail. The cut is commenced at the top of the coal and a groove as wide as the machine will allow and as deep as it will reach is finished, after which it is lowered for successive

FIG. 320.—A Coal-shearing Machine.

and adjoining cuts. The cutters travel downward along the coal, thus always drawing the chippings down and out.

Many of these coal-cutters are made adjustable in position for undercutting or shearing. The latter is not an economical method of procedure as compared with undercutting, but is serviceable in firm coals and gaseous mines where it is advisable to use powder.

Another shearing-machine, made by the "Sullivan Machine Company," is of the pick type, mounted on a truck and pivoted to swing up and down to the point. Its usefulness, however, is confined to the centre of headings and it cannot be employed for shearing ribs. The machine makes a cut 6 inches wide and from 7 to 8 feet deep, averaging 30 lineal feet in a ten-hour shift.

The Comparative Advantages of Coal-cutting Machines.—Coal-cutting machines can be employed to advantage underground when the coal is firm and in seams which are of standard dimensions, free from eccentricities, and under a strong roof with a good floor. They are employed when it is necessary to develop the colliery rapidly and where it is essential to obtain a maximum proportion of large coal. They have an average over handwork in concentrating the field of operations. They require less men and are especially advantageous where shooting off the solid rock is practised and where there is no soft underlying layer in which the miner can underhole the coal. They cannot be employed to advantage in wet mines, where the work is intermittent, where the pressure of the roof bears too heavy on the coal face, or where the layer under the coal is pyritiferous, or in the anthracite coal, which is full of slip-joints.

The statistics furnish ample evidence that the powder consumption in machine mines is less per ton of coal than in hand mines. Likewise the number of fatal and serious accidents occurring in machine mines is much less. It usually requires from three to five additional machines to maintain an uninterrupted work for seven machines.

A machine can cut a groove of 6 feet depth and 44 inches width in from 3½ to 5 minutes, and with the time occupied in making six changes the undercutting of a room 20 feet wide can be effected in two hours' time. Assuming thirty minutes as necessary to shift the machine to the next room and to reset it, it will be possible to underhole four rooms in a day with the employment of two men. This in a standard seam of 4 feet thickness corresponds to 144 tons. By handwork with two men at a face, eleven rooms would have to be kept open to equal the supply of this one machine, which operates in four rooms. A mine producing 1000 tons of screened coal, 1300 tons of run-of-mine coal, can obtain it by nine machines in thirty-six rooms or by 200 miners in 100 rooms. The operations therefore are more concentrated, as less territory is kept open than in the handwork, and this is an important feature when there is taken

into account the maintenance of roads, ventilation, and supply of timber.

One marked objection to coal-cutting machines lies in the quality of coal which is produced. This arises from the fact that the coal is broken in large lumps, which contain an undue proportion of impurities. They require less explosive, and indeed necessitate the use of light charges, since heavy shot-firing weakens the pillars and wastes the coal. Cutting-machines require broad rooms to obtain the maximum quantity of coal at the cheapest rate, and this results in the reduction in the size of the pillars, with the consequent sacrifice of much coal in the ribs and stumps. Moreover its accumulation in the gob tends to develop spontaneous combustion.

Percussion Machines versus Chain Machines.—The former class of undercutting machine is especially advantageous over the other types when the roof is so weak as to require timbering close to the face and where the bearing-in layer under the coal is too stony or too full of pyrites to be cut with continuous-motion cutters. It has also an advantage where there is excessive pressure from the roof upon the face of coal. It reduces the cost of mining to quite a considerable degree even when the cost of operation, interest, and maintenance are included. It is, however, slow and produces a great deal of slack coal, besides doing less work in a given time than can the breast machines. The latter can undercut in fire-clay or in soft under layer quicker than the punching-machine. In fire-clay the chain machine is preferred, while for soft coal or slate the stronger disc or wheel must be used.

The breast and longwall machines, employing chains, wheels, or rotating bars, can be operated by electricity or by air, whereas the reciprocating machine must be operated by compressed air. The cost of installing their power plants is about the same for an equipment of equal capacity and a distance of one mile from the power-house. For greater distances electricity admitted is far cheaper than compressed air. In comparing the cost accounts of producing coal by either type of machine not

only are wages, materials, interest, and depreciation to be considered, but also the profit on the amount of coal lost by each method.

Comparison of Hand-mining with Machine-work. — The amount of slack dirt and waste which must be allowed for in hand-mining is as a rule 50 per cent more than with machine.

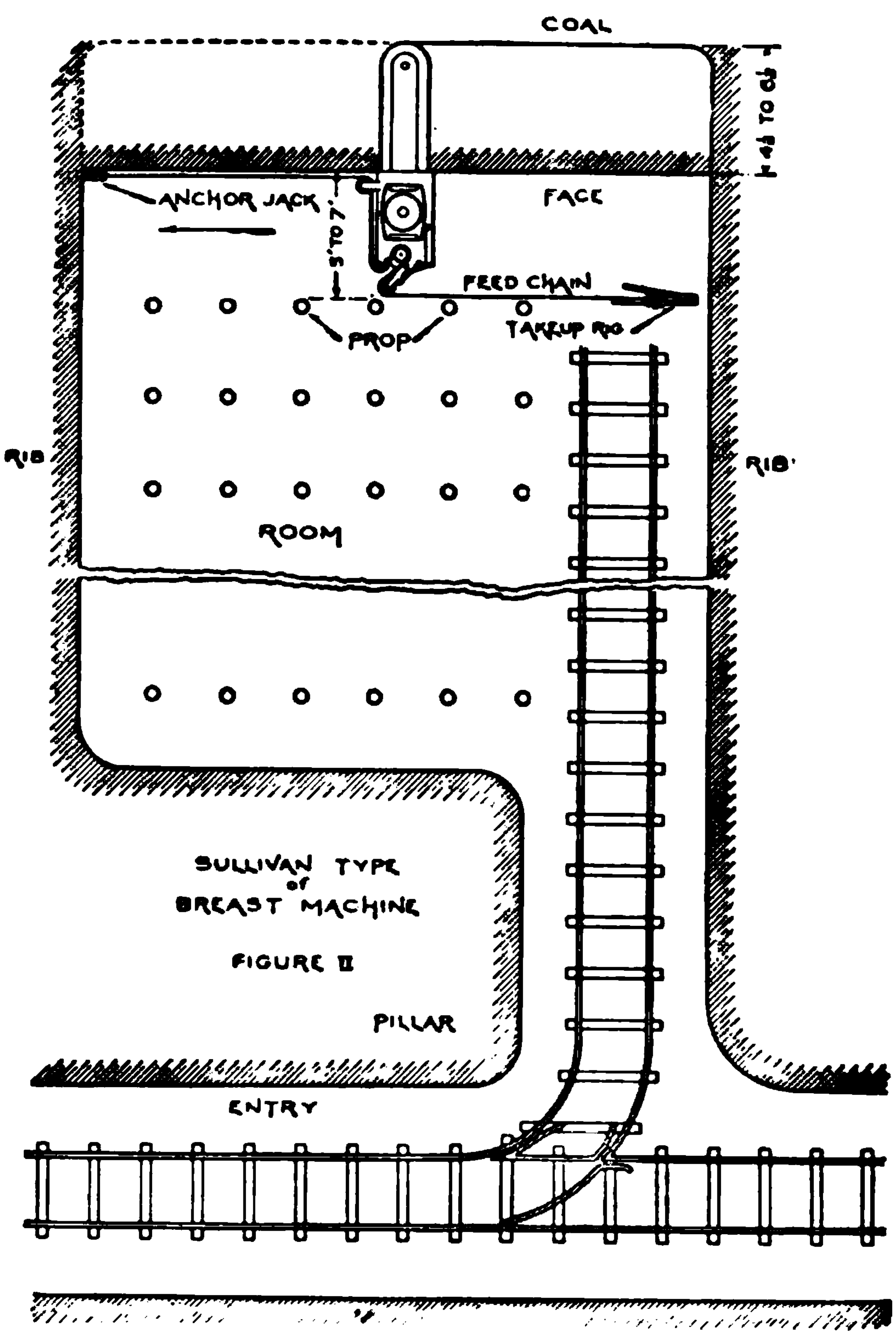

FIG. 321.—Showing Position of Sullivan Coal-cutter in a Room.

When the net profit it represents is included the saving of machine coal over hand-mined coal is still greater. The machine-cut groove 6 feet deep saves three cubic feet of coal per yard of face in the reduced slack. A machine cannot be employed to advantage in seams as thin as those which can be mined by hand. Neither can the breast or longwall machine be employed in as

narrow and confined quarters as can a percussive-machine cutter. As to output per year 13,000 tons may be cited as an average production for either class of machine. Hand labor as compared with machine is expensive, and the tendency therefore is to confine the former to narrow work and the extraction of coal from the ribs and pillars. At the present time more than one quarter of the soft-coal product of the United States is being machined, with a rapid increase in its adoption with time. When hand labor can no longer be employed profitably the ribs and pillars must be recovered in the second or room working by some other type of machine than those employed at the present time. However rapid the present machine may undercut or shear the coal it is not sufficiently rapid for the extraction of the rib coal in its entirety without considerable loss and risk. The longwall type of machine will probably be called into requisition even though it may differ from the character of the machine employed in the rooms and workings of the mine. In this regard the Sullivan coal-cutter, Fig. 318, meets the requirement, as it may be moved parallel to the face by a feed-chain, as shown in Fig. 321.

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CHAPTER VIII.

BLASTING.

The Principles of Blasting.—The principle employed in rupturing rock consists in subjecting the surface of a subfacial cavity regular or irregular to a sudden increase of pressure, acting radially outward. When the agent is sufficiently powerful to produce a high degree of compression upon the surrounding rock it either fractures the material by the formation of a congeries of crevices or it shatters it. The extent of the destruction depends upon the intensity of the pressure and the cohesion or the toughness of the material.

Blasting-agents.—There are two classes of blasting-agents. The first includes all varied means of causing a prolonged pressure of greater or less intensity, while the second develops the force more or less instantaneously. In the first class are mechanical devices such as the steady hydraulic pressure obtained from a ram; that of pistons actuated by compressed air; the swelling of slaking lime; and the spreading produced by wedges. The second class comprises various chemical combinations such as black powder, nitroglycerine, ammonite, etc.

Blasting-agents of the First Class.—These are employed for driving preparatory works in the fiery mines and are safer and less expensive than powder. Lime is a safe effective blasting-agent in coal, and can be made up into cartridges, which after insertion into a deep hole are packed tight and moistened. In a short time the lime slakes and expands to break the coal away. Compressed air forced into a cylindrical case by a powerful air-pump is capable of exercising such a pressure as may be sufficient

to break the coal. This is employed in some coal regions and is a simple, safe, cheap method. Wedges driven by any of these agents are similarly employed by being inserted in the line of the cleavage of the rock.

Explosives.—An explosive, according to André, is a mixture "capable of being suddenly transformed into gases by the application of heat." In this sudden evolution of gas, in a space formerly occupied by a solid, a pressure is produced upon the confining surface to the volume of the evolved gas to that of the explosive. The expansive force of the gas is greater as the temperature of ignition increases and with the rapidity of evolution of the gas. If the latter was instantaneous, the maximum pressure is imparted at the moment of explosion; if the combustion is slow and transmitted from grain to grain, its strength is dissipated over a longer period of time, and the pressure is less. Thus the strength of an explosive is measured by its specific volume, the amount of gas it produces, the temperature, and the rapidity of the evolution.

Below is a table of the relative volumes and pressures of gas produced by the perfect combustion of 1 lb. of explosive occupying about 0.016 cubic foot of volume.

	Volume of Gas.	Heat-units.	Relative Force	
			By Single Explosion.	By Detonation.
Blasting-powder	2.38 cu. ft.	900,000	1.00	4.34
Chloride of nitrogen	5.09 "	570,000	1.08	3.61
Guncotton	11.01 "	1,050,000	3.06	6.46
Picric acid	10.72 "	1,240,000	3.15	6.00
Nitroglycerine	9.76 "	2,375,000	4.55	10.00

When an explosive is ignited the heat developed in its combustion is rapidly communicated from grain to grain and the particles decomposed with the liberation of gases. According as these two phenomena follow each other slowly or quickly we have rending or shattering powders, of which the representa-

tive types are black powder and nitroglycerine. In the first the gas is evolved so slowly as to give time for a concentration of pressure along a line or lines of least resistance. This is the quality desired for the artilleryman or for a sporting powder—ability to project. The slow combustion operates upon the small mass of the bullet, which can instantly take up a very high velocity and thereby give a rapidly increasing space for the evolved gases to escape. If the bullet or plug were tight and moved slowly it would burst the breech or muzzle.

The miner, however, desires to *break*, and this property is obtained from such agents as rapidly produce gases at high initial temperature and pressure. Very little plugging is needed, for the concussion produced by the gases of the quick powder is practically instantaneous and the wave of pressure extends in all directions, which, being resisted by the rock, spends its force there and shatters it. The more sudden the action the more local is the effect.

Igniting and Detonating Explosives.—Without attempting to follow the history of blasting, for which the student is referred to Rziha's "Lehrbuch der Gesammten Tunnelbaukunst," an enumeration of the several simple and compound substances used or suggested at various times to produce concussion may be here given in chronological order: Common black powder, picric acid, guncotton, terchloride of nitrogen, nitroglycerine, and ammonite. This list may seem brief, but a longer list would be merely an enumeration of the varieties obtained by the substitution of a single constituent. We have the artillery, sporting, and blasting-powders, composed of charcoal, sulphur, and saltpetre in varying proportions; picric acid and picrates with saltpetre or chlorate of potash; guncotton, combined with other explosives; nitroglycerine, with admixture of absorbents and dilutants. The result is that we have various grades of explosive compounds, from those which may be ignited by heating to a temperature of about 300° C. to the nitroglycerine, which requires a shock. In other words, we have igniting or detonating compounds.

Black Powder.—The term black powder, or gunpowder, embraces mechanical mixtures of carbon, sulphur, saltpetre varying from 12, 8, and 80 per cent respectively for sporting purposes to 20, 16, and 64 per cent for open-air blasting and 11.5, 17.5, 71 for hard-rock underground. It is black or brown according to the per cent of carbon. Its constituents are pulverized, compressed into a cake, granulated, sifted, glazed, and dried. The size of the grains depends upon the use to which the powder is to be put. The smaller-grained powders are preferred for mining purposes. The color of the powder varies from black to dark brown, the former having a larger percentage of carbon than the latter.

Methods of Charging the Hole with Powder.—When the hole has been drilled to the required depth, the powder is either poured into it from the can or is inserted as a cartridge. The cartridge is a tube of oiled paper which is closed at the bottom and also at the top after the black powder and fuse have been inserted.

Black powder is loaded by the barrel method or the needle method. In the latter method, which is more common, the cartridge is inserted into position with a needle projecting from it out to the daylight. Above the powder, around the needle, and filling the hole for some distance is driven a very soft-clay packing material, which is rammed tight into position. The needle is then withdrawn and replaced by a fuse for ignition. The needle is a slender-pointed rod of copper with a ring at the upper end.

In the barrel method the powder cartridge is pierced by a wire which leads up through a half-inch copper tubing, or "barrel," that extends the entire length of the hole. Around the barrel is tamped the packing, after which the wire is pulled out and replaced by a fuse. A comparison of the methods shows a preference in favor of the barrel because less and poorer quality of tamping may be used; it is twice as fast; the cartridge has less opportunity to soak water; and the cheap barrels are recovered after shooting. Some anthracite miners use a $\frac{3}{8}$-inch wrought-iron pipe for the blasting bale.

A Blast-hole Loader on the Barrel System is in use for loading and tamping large charges of powder without waste or danger of premature explosion. Tubes in 5-foot lengths are screwed together with a funnel at the top, and a short bronze discharge-tube is at the lower end. A hickory tamping-bar completes the outfit. A system so complete and compact saves time and powder (Fig. 340).

Tamping and Ignition of the Powder.—To confine the large volumes of gases within the small volume occupied by the powder a tight cover of melted clay is rammed on top of the charge and packed tight. The process is known as tamping.

As a precaution to prevent a premature explosion the material used must be free of any hard substances which may produce sparks by accidental contact with the tools used in the process. The rammer should not be of metal. A stout hickory bar is usually the tamping-rod. Frequently tamping-bags of stout paper are filled with the tamping material, when the latter is of low quality and contains grit. This prevents contact of the gritty substance with the rock surrounding the hole. Anthracite miners use a half-inch iron bar clubbed at the end to 1½ inches for a tamper. This has a groove at the side to allow for the needle.

For the same reason the needle mentioned above must not be of iron or steel, but of copper. Sometimes it is merely a copper-tipped iron rod. In either case it is sharply pointed and when withdrawn leaves a cavity for the insertion of the fuse. Black powder is ignited by intense heat or by a spark or by detonation. The last named is produced by the explosion of a small quantity of fulminate of mercury in a cap by conducting to it sufficient amount of heat through a fuse ignited at the surface. The detonator may be dispensed with and the heat from the fuse alone may be sufficient to ignite the powder. The result is not so beneficial as when obtained by detonation, as shown by the table on page 672. An electric spark or detonator may also be employed.

When the hole to be fired is in creviced rock, which is liable

to dissipate the force of the gases, a "bulling-bar" or "clay iron" is used before charging the hole to stuff soft clay into the crevices.

The Fuse.—The fuse, also called a squib, is a thread of powder wrapped in tarred hemp or in cotton and water-proofed outside. When properly made it burns uniformly at a rate definitely known to the miner. One variety, for example, burns at about 20 inches per minute. A knowledge of the rate enables the miner to determine how short a length of fuse may be used and still provide him with safe escape. The fuse is supplied in rolls containing 24 to 40 feet and is cut off in lengths as desired. It is inserted into the cartridge and lowered into the hole with it or it may be used in connection with the needle as previously described. It furnishes a means of communicating the necessary spark and heat for the ignition of the powder and frequently is fitted with a fulminating cap at its end.

The Cap. — A fulminating cap is of copper $\frac{5}{8}$ inch long and $\frac{1}{8}$ inch diameter, having a small quantity of fulminate of mercury deposited in it. This chemical explodes with heat with violent fuse and ensures by its detonation the prompt ignition of the powder in the cartridge into which it is inserted. The cap is slipped over the end of the fuse, the top of it being greased with a little cartridge soap. Numerous accidents have happened from careless handling of the caps which contain the violent explosive, and care should be taken that they do not fall into the hands of the careless miner without suitable precaution being exercised. The detonation of the explosive in the cap increases the effect of the black powder fourfold beyond that obtained by a mere ignition with the use of the simple fuse, as may be noticed by the table given above.

Precautions in the Handling of Black Powder may be summarized as follows:

1. That black powder and high explosives of any kind are not safe to use together in the same hole.

2. The cap is the only exploder that is safe to use in firing off high explosives, which does away entirely with tamping.

3. The only tamping necessary on high explosives is a small

piece of miners' clay, which is easily put down on the charge with a wooden bar.

4. The practice of picking and boring out missed holes that have been charged with either black powder or high explosives should be strictly prohibited.

5. Any miner on the mine known to use black powder and high explosives together, thereby necessitating tamping, will be discharged.

The gases produced by the explosion of the fulminating powder have the following percentages: CO_2, 43; N, 35; CO, 12; H, 6; carbohydrates, 4.

Black powder is employed for all varieties of rocks and is very serviceable in coal. Its service in the Galena veins and soft rock is better than obtained from the more violent shattering explosives, since it does not pulverize the mineral so finely.

Lewising.—Granite is often quarried by a method known as lewising. Several holes are drilled close together and the partitions between them broken down with a flat steel bar, or broach-bit (Fig. 299). This extensive hole fixes the direction of the fracture, which is usually selected as parallel to the "rift" or cleavage. Three drill-holes make a "complex" lewis-hole. The benefits of this lewising may also be secured by the Knox system, which is meeting favor for dimension work. The hole having been drilled, a reamer cuts V-shaped grooves in its opposite sides to determine the line of break. The tamping is not driven down on the powder, but an air space is left between them. This scheme permits expansion of the gases and gives time to effect rupture along the plane desired.

Powder Consumption.—The amount of powder consumed for a given work varies too greatly for any general statement. One foot of a 1-inch hole is capable of containing 5 ounces of black powder or 38 inches for a pound. The powder, to be economically used, should not fill more than one third of the hole. In anthracite mines a keg of powder weighing 25 lbs. is consumed for every 40 tons of coal. The bituminous miner breaks on the average 300 tons with a keg. The amount of powder

consumed per ton of coal worked by hand and that by machine is about the same. In longwall mining the consumption of powder is very small.

Nitroglycerine. — Chemically known as trinitroglycerine, or glonoin oil, $C_6H_5N_3O_{18}$. Made by treating glycerine to nitric and sulphuric acids at a low temperature. The temperature at which it will fire is 360°. It is exceedingly sensitive to shock and will explode under the influence of a neighboring blast. It will not take fire when touched by a red-hot body, or if it does, it burns quietly without smoke. If a thin layer of it be struck by a severe blow it will instantly explode. If heated to the temperature of combustion or detonated or even jarred by a wave of concussion its liquid mass will be converted at once into gas and if confined will burst its casement. The process of decomposition will be complete and a perfect combustion of the chemical elements will ensue. A small amount of it explodes by detonation, even if not buried in the rock, but when lying on the face of a boulder in the open air, the surrounding air will not be able to move aside, or to take up the rate of vibration of the shock quickly enough to allow it to expend its energy on the air and the rock underneath will be badly shattered.

The Manufacture of Nitroglycerine.—Several jars of one-gallon size are placed in a cool stream. Into them are poured nitric acid and sulphuric acid in due proportions. A fine stream of pure colorless glycerine is then poured on while stirring the mass. When boiling begins thick red vapors of nitrous gases are emitted. If the glycerine is poured in too fast, or the water is not sufficiently cool, the mixture will take fire with a hissing noise, and if it be not stirred enough the blaze will shoot up several feet. Stirring is continued until the action becomes less intense. Each jar in turn receives a small charge of glycerine, after which the process is repeated. When no more fumes are given off and the addition of glycerine produces no effect, the decomposition is complete and a heavy milky, oily fluid will be noticed on the bottom. The acids are decanted and washed thoroughly with water. Into an old-fashioned wooden churn the nitroglycerine

is then poured with an abundant supply of water and churned actively until litmus paper shows no acid reaction. Upon the freedom from all acids depends the safety and the permanence of the nitroglycerine. The product being poured into a wooden bucket with some clear water its milkiness will soon disappear and a colorless compound is the result. If after a few days it discolors the acid must be removed. A yellow tinge requires the additional churning with cold water; an orange color requires some alkali; but if it becomes cloudy and deeper in tinge the material is valueless and should be either discarded or replaced at once.

Dynamite. — This explosive was discovered in 1864 by M. Nobel and has received universal adoption in a variety of forms for almost all purposes of blasting. The term is a generic one and includes any mixture of nitroglycerine with some inert or mechanical absorbent material. Earth, sawdust, wood pulp, infusorial earth, and magnesia are used as absorbents, in addition to which are added in some cases explosive bases which increase the strength. Nitroglycerine is the active principle of the explosive and its percentage in the compound determines the power of the explosive. Infusorial earth, which absorbs three times its bulk of nitroglycerine, excels all other bases and furnishes a very strong explosive. It is usually graded as No. 1. The weaker grades of dynamite are designated as No. 2, No. 3, etc. The strength of No. 1 is about three quarters that of the pure article and six times that of black powder. The powders known as Atlas, dualine, fórcite, etc., differ only in the nature of the material used for absorbent.

The compound is like moist brown sugar in color. It freezes at 46° F., hardening into a white mass. It is almost as sensitive to sudden changes of temperature or pressure as nitroglycerine. It is perfectly safe to handle and is harmless so long as it does not exude from its absorbent.

The constituents of some of the high explosives are indicated below:

Tonite is macerated guncotton 52.5 and baryta nitrate 47.5.
Gelatine is soluble guncotton 2.5 and nitro 97.5.
Dualine is nitro 50, sawdust 30, and nitre 20.
Rendrock is nitro 40, paraffine 7, nitre 40, and wood fibre 13.
Atlas A is nitro 75, fibreless wood 21, nitre 2, and magnesia 2.
Hercules No. 1 is nitro 75, chlorate of potash 1, nitre 2, sugar 2, and magnesia 20.
Giant No. 2 is nitro 40, rosin 6, sulphur 6, absorbent 8, and nitre 40.
Rackarock is nitrobenzol 22.3 and chlorate potash 77.7.
Vulcanite is mealed gunpowder and nitro in different proportions.

Experiments upon the relative efficiency of the various explosives have been made under water and are recorded in General Abbot's "Submarine Mines," but no formulæ can be prepared from the results because of the varying influences of the foreign substances added to the main explosive.

The Storage of Nitroglycerine. — Nitroglycerine and its various compounds may be exploded by sudden increase in the temperature and some by a neighboring shock. All are liable to decompose under a low heat. It will exude from its absorbent when hot or exposed to a wet atmosphere. Precaution must therefore be taken in the storage and the handling of the material to avoid these sources of danger. They should be stored in large cool dry caves or sheds well provided with ventilating flues and remote from scenes of blasting. They should not be allowed to freeze, for, while in that state they cannot be fired, thawing will be necessary before they can be of service. The thawing of dynamite introduces an element of danger, unless care be taken to avoid a sudden increase in temperature or an excess of heat. Dynamite is thawed out by slow heating in a bucket immersed in another. In a jacket is circulated lukewarm water. If the bucket be kept away from the direct fire or heat an excessive heating is impossible. Dynamite which has been stored a long time should be tested by litmus paper for acid reaction, which indicates decomposition, from which spontaneous combustion may ensue. Less accidents occur from the storage of dynamite or nitroglycerine than from the keeping of black powder.

Being a more powerful mixture than black powder, the nitroglycerine compounds are more economical by reducing the labor

of the miner; requiring less tamping; breaking the mineral finer; requiring smaller holes; consuming less steel and supplies, and materially shortening the time of mining. As water does not injure it, it is without rival as an explosive in wet ground, and this becomes more manifest as the rock encountered is harder.

Methods of Charging with Nitroglycerine.—The holes are loaded with nitroglycerine by pouring from a tin cup upon the fuse with its cap and covering the mass with water. Dynamite may be loaded like black powder, the cartridges into which it is made being obtainable of any desired diameter and of a length of about 8 inches each. As many such cartridges may be inserted as necessary. Safety-fuses or electric wire and cap are buried in the uppermost cartridge, which, after placing in position, is slightly tamped with earth and ready for ignition.

Black powder is often used in combination with giant powder "to start the hole." The proportion is usually one half pound of black powder to three sticks of giant. This practice is often questioned, but the fact that the entire ground is broken to the very bottom of the hole without leaving any collar at the surface is sufficient evidence of its advantage.

The Secondary Explosion of Powder.—The great disadvantage attending the use of any form of explosive, particularly in coal-mines, lies in the production of smoke or gases more or less noxious. The combustion of carbon hydrogen and sulphur in the presence of the oxygen provided by the various nitrates added to the explosive results in the production of carbonic acid, carbonic oxide, sulphurous acid, nitrous, and other injurious gases, and water, which necessitate plentiful ventilation. When the combustion is perfect and complete all the carbon is converted to carbonic acid and the sulphur to sulphuric acid. But in the deficiency of oxygen resulting from the addition of adulterants or an imperfect mechanical mixture of the constituents, carbonic oxide is developed as well as smoke, which, with the other injurious and combustible gases, are injected from the hole into the room or stope. Sometimes this is accompanied with a large volume of sparks. The conditions are therefore favorable

for a secondary explosion in the room, which might be further aggravated by the presence of coal-dust or gas, and thus form a nucleus for a bigger explosion after being brought into contact with neighboring volumes of gas. For this reason common black powder, blasting gelatine, and carbonite are prohibited in many coal-mines inclined to be dusty or fiery. Every commissioner who has examined into the safety and efficiency of blasting-powder has reported that black powders are dangerous and should be prohibited. Some of them are so highly adulterated that not only is the combustion incomplete, but the gases evolved by the combustion afterwards dissociate and sometimes recombine, forming dangerous gases that may incite secondary explosions. Nitroglycerine and dynamite, which are chemical compounds capable of complete and instantaneous combustion, are much safer, besides being more economical, since no carbonic oxide is produced. These latter explosives may be employed to great advantage and with less risk even in the dry, dusty, and fiery mines.

Flameless Explosives.—A variety of explosives have been made which produce no carbonic oxide and hence no flame, and which are known as *safety explosives*, sometimes also called Sprengel explosives. These are composed of a mixture either of two solids, a solid and a liquid, or two liquids, one of which should be a hydrocarbon and the other a compound rich in oxygen, neither at the same time being sensitive to friction. Principal among these is a group of explosives—ammonite, roburite, carbodynamite, and bellite—in which the chemical to supply the oxygen is a nitrate, as in black powder. Nitrate of ammonia or of barium was most commonly employed, while the other combustible is nitronaphthalene or dinitrobenzol. Owing to the deliquescent nature of ammonium nitrate the two elements are not mixed until ready for immediate use. When so mixed the explosive is dipped in melted wax to keep it dry.

A detonating wave such as is produced by the fulmination of mercury is alone capable of firing it. When the two elements are intimately mixed in proper proportions, the shock of the

detonation is communicated to the layers of molecules in the immediate proximity of the cap and the powder, whereby the "molecule edifice" is destroyed. This initial force is augmented to the degree corresponding to the heat evolved in the decomposition until the total is consumed. The decomposition of the ammonia nitrate absorbs such an amount of heat from that of the initial force as to reduce the final temperature of the gases developed and projected into the air below the temperature of ignition of fire-damp. Owing to the impossibility of an incomplete combustion, an excess of available oxygen is added in an excess of nitrate of ammonia, which, absorbing more heat, effectually prevents the formation of carbonic oxide and nitrogen oxides. Changes of temperature do not affect the mixture, freezing for three months does not injure it, and there is no exudation to endanger it. Unlike gunpowder or any nitroglycerine compounds, it will not explode by percussion, fire, or electric spark. If struck with a heavy hammer the portion of the explosive directly hit is decomposed by the heat developed by the blow, but the remainder is not affected. If mixed with gunpowder and fired the latter explodes, scattering the former without affecting it.

Detonators.—These are required to produce a violent concussion and an explosion, if otherwise weak explosives, and are of different grades, depending upon the quantity of fulminate of mercury in them. The strongest contains 23 grains of fulminate and is known as No. 1; No. 2 contains 15 grains, and No. 3, 8 grains. The strength of the detonator used increases with the uncertainties of the two constituents of the powder. Ammonite containing between 87 and 89 parts of ammonium nitrate and 11 to 13 parts of dinitronaphthalene, and bellite which contains 79 to 81 parts of nitrate with 19 to 21 parts of metadinitrobenzol, require a No. 1 detonator. Roburite with nearly the same amount of nitrate, but containing the more sensitive chloronaphthalene and dinitrobenzol, and carbonites containing 30 to 36 parts of barium nitrate, 25 to 27 parts of nitroglycerine, and 37 to 43 parts of wood meal, may be fired by a No. 2 detonator. Ardeer powder, containing a little more

nitroglycerine than carbonite, with 11 to 13 parts of kieselguhr, requires for its explosion a No. 3 detonator.

These detonators add a new element of danger to mining, for in their deflagration a spluttering stream of sparks frequently results, which is liable to ignite any coal-dust which may happen to be present in the room. This is the main objection to the use of flameless explosives, though the latter are safer than the other forms of powder now in use. Perhaps the use of a detonator of still higher power might reduce the danger by producing total combustion, but it is probable also that the increased temperature and the heat evolved might be sufficient to ignite the gaseous mixture in the room.

Safety-explosives cost nearly twice that of black powder, besides producing a greater proportion of coal-dust.

Smokeless Powders.—Powders which are said to be smokeless are made by mixing in suitable proportions guncotton and nitroglycerine, their combinations being effected by the use of camphor and acetones. Their energy is moderated by the addition of inert materials or by increasing the proportion of guncotton, though this results in a lower rapidity of explosion. Such powders are handled in small cubical grains and are safe. They are exploded by detonation, not by percussion.

"Schneiderite" is a powder, light yellow in color, quite oily to the touch, forming lumps readily when pressure is applied. Considered alone "schneiderite" is a wholly inert substance of perfect stability and containing in itself no explosive substance whatever. The elements of which it is composed only combine to form an explosive at the very moment of the explosion under the influence of a detonating primer.

When the detonator is not used "schneiderite" may be submitted to the most violent shocks with impunity. It is not influenced by fire. Thrust into a fire it burns with difficulty. It is also uninfluenced by the most extreme cold. It is sensitive to but one alteration and that only diminishes its explosive qualities; this is the alteration which may result from its hygroscopicity. To avoid the absorption of moisture it is necessary to make sure

of the imperviousness of the cartridge papers. It is easy to restore all its properties by drying it in a stove or simply in the sun.

The handling or the transportation of "schneiderite" is not dangerous under any circumstances or under any conditions of preservation.

The Theory of Blasting.—When an explosive is fired, the tension of its gases acts uniformly in all directions upon its encasement. A tendency to rupture is developed along the line offering the least resistance to the action of the powder. In a rifle the bullet finds a comparatively easy line of escape along the barrel. This is the line of least resistance. In rock the line of least resistance will be along one of the systems of cleavage planes, in the direction of the shortest distance from the centre of gravity to the surface or along the bore-hole itself.

It is the purpose of the miner to ascertain the direction of the line of least resistance and to drill the bore-hole to be loaded with powder in a direction approximately at right angles to it. Except in massive rock presenting a plane surface for attack this line of fracture cannot be arbitrarily selected, for the powder will seek its easiest vent for escape from the explosive chamber. For this reason a hole cannot be drilled along the line of least resistance. It becomes merely a cannon; the powder blows out without other result.

The Line of Least Resistance.—In Drinker's "Explosive Compound" will be found a table showing the relative resistances of different rocks and coefficients representing their toughness. By its use may be determined the quantity of a given grade of powder necessary to produce the rupturing effect. Let W be the weight in ounces of the blasting-agent, L the distance in feet of its centre of gravity from the surface of the working-face, and C the coefficient of the rock resistance, then $W=CL^3$. In a given mine the value of C may be determined by a series of trials, the other two variables being altered in amount, employing as little explosive and placing the hole for as long a line L as possible. Thus if a 27-inch hole breaking rock along a line of

20 inches length shows an average consumption of 5 ounces dynamite, then C becomes 1.09. A subsequent hole with a line of least resistance of 30 inches would require 17 ounces of the same explosive.

The line of least resistance, which is the line of general throw, extends from a point approximately the centre of the charge to the nearest external point, from *a* to *b* in Figs. 322 to 325.

FIG. 322. FIG. 323. FIG. 324. FIG. 325.

Illustrating the Line of Least Resistance.

In soft rock whose cohesion is comparatively slight, the line of least resistance may be a long one—nearly that of the depth of the hole (Fig. 322). In tough rock the line *ab* is short (Fig. 323). The volume of rock broken down is proportional to the weight of explosive used and to the cube of the line of least resistance. With a given depth of hole and of charge in it the length of line is fixed. The volume of rock broken is determined. To increase the effectiveness of the powder the line must be increased by varying the direction of holes or the amount of powder diminished. A deep hole loaded with a powerful explosive is hence more economical than shallow holes or low grade of powder.

FIG. 326.—Shooting in Tight Ground.

When the line of least resistance exceeds in length that of the hole, as in Fig. 326, no execution is done, other than that

indicated by the result at *K*. The inclination of the drill-hole with a flush working-face cannot ordinarily exceed 45° when black powder is used. It may be deflected to 60° if the rock is soft and the explosive powerful.

Blasting in Creviced Rocks.—The cleavage planes of the stratified rocks furnish numerous lines of least resistance which may be advantageously employed as lines of rupture. Wedges or picks are often inserted to split them. Certain coals break freely along these cleavage planes. In some of the massive rocks the rifts and seams are sufficiently pronounced to afford lines of weak resistance to rupture. The miner avails himself of these faults in the rock and drills the hole normal to them but not down to the seams in Fig. 327. If the centre of gravity of the powder is well below the crevice *b*, the block above *a* will be loosened for a considerable distance on either side of the hole.

FIG. 327. FIG. 328.

Drilling in Creviced Rock.

In the case illustrated in Fig. 328 the line of least resistance is across to the free face of rock, but there is likewise a tendency to split along the plane *b*; the rock will doubtless break downward to the right. In Fig. 261 the shot *m* has broken to the bedding plane, and somewhat shattered the rock ahead. The shot *n* and that at *m* (Fig. 262) will break to the black clay gouge. That at *o* will probably blow out without any execution. The shot *p* will blow out a large block if those near *o* have been properly fired. By firing the holes in order from above downward greater extension is effected, as the bedding planes assist the miner. The shot *g* in Fig. 263 is well placed to break the rock at the plane of cleavage. With the stratification dipping down from the face the shots proceed in order upward.

In coal and soft or brittle metalliferous minerals a weak explosive is used to avoid injury to the product. When a thin vein of the latter adjoins a hard rock the usual practice is to blow down the rock with lightly loaded holes. In shafts or tunnels

no attention is paid to the condition in which the mineral is broken down, as rapid progress is sought. For simultaneous firing no heed is paid to the economy of the line of least resistance. Such holes are machine-drilled and are limited in inclination.

Blasting in Massive Rocks.—In homogeneous massive rocks there is no pronounced crevice or seam to assist the miner in the displacement of large masses. The line of least resistance must be created by properly directing the drill to blow down the maximum possible volume of rock with the minimum of drilling. But an additional problem is presented. The miner must consider the manner in which working-face will be left after the shot. Each shot in turn becomes a "bearing-in" shot for its successor (Fig. 329). Without any planes of cleavage to which to break porphyry, vein quartz, and granites, the direction of the shooting must be to their working-face. This problem is of special difficulty when the working-face is the small area of a tunnel or shaft. Such an explosive is known in miners' parlance as "tight ground." Considerable skill is necessary to accomplish much work under such circumstances. The usual practice consists in so breaking the ground as to avoid plane surfaces, but produce irregularities of profie such as are illustrated in Figs. 324 and 330 to 332. Rupture

FIG. 329.—Bearing-in Shots.

FIG. 330.

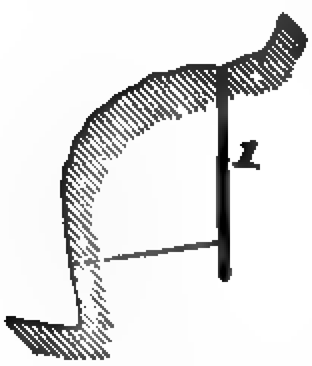


FIG. 331.

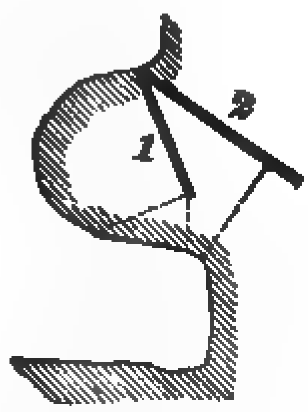
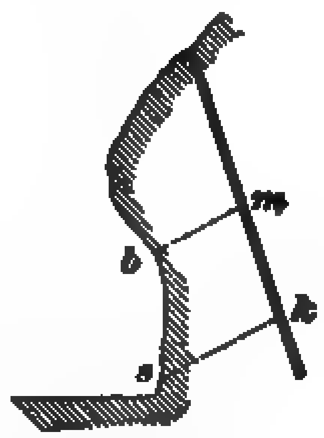


FIG. 332.



Taking Advantage of Irregularities of Face.

may then be effected along the dotted lines toward the recesses in the rock. Judgment must be exercised in directing the holes;

as, for example, hole 2 in Fig. 301 will displace more rock than No. 1 with equal powder and work. Indeed it may happen that the powder will simply pulverize the rock along the line of break of No. 1, dislodging the bowlder alone. So in Fig. 260 is illustrated a poorly directed hole. If drilled to *k* the blast will take place along *ks*, but only a small charge is needed for the short line. As drilling is an expensive operation it would have been more economical to have drilled it less vertically. Had the operations stopped near *m* no more powder would be required to blow out at *bm* than at *sk*, though difficulty would have been experienced later in blasting out *bmks*.

Expanding-bits.—A soft or brittle rock may be blown out in large masses by excavating a large chamber at the bottom of a

FIG. 333. FIG. 334.

An Expanding-bit.

very deep hole and filling it with black powder, sufficient for the work. The chamber may be produced by an expanding-bit

(Fig. 333) or by a small charge of powder. When the desired depth of hole has been reached a pair of cutter-wings are forced out, which when revolved cut out a chamber. Powder accomplishes the same end by shattering the rock. This method is known as squibbing.

Simultaneous Firing.—It has been shown that increased effectiveness is obtained from an explosive by the use of perfect detonators. A still further gain results from the ignition of the shots simultaneously, by which they materially assist one another in effecting disruption. The concussion produced by the explosive, though progressing radially in every direction from the centre of the charge, succeeds in rupturing the rock only over a limited volume, as is represented in Fig. 335.

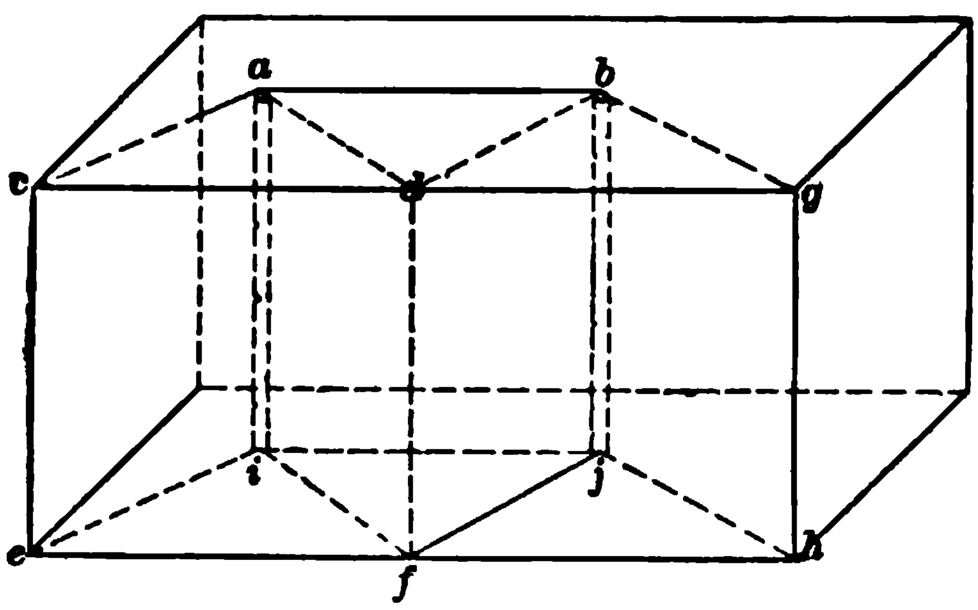


FIG. 335.

Either shot (*i* or *j*) drilled into the block would break out only a triangular prism. The hole *i* would break the prism bounded by the parallel faces *CAd* and *eif*, or the hole *j* would break the prism bounded by the faces *ABD* and *IFJ*. The rock is broken toward the free face, but the force expanding laterally and backward into the solid ground is entirely lost, the amount lost increasing with the intensity of the explosion. If, however, several contiguous holes be so placed that their lateral rupturing tendencies will superpose, they will shatter rock which by the single shooting would only have been fractured. The holes *i* and *j* would break the mass of rock included between the trapezoids *ABGC* and *IJHE*. The latter mass is manifestly much in excess of the aggregate product of the single holes.

The zone of an explosive effect may be divided into three sections—a sphere of pulverization immediately surrounding the bottom of the hole, a cone of rupture with its apex near the centre of gravity of the cartridge and its base of an area dependent upon the relative explosive and resisting powers, and a zone of vibrations extending into the rock where the undulations are too weak to produce more than an occasional fracture. By simultaneous firing, the explosives may be placed at such distances apart as to permit the overlapping of the zones of vibration, whose cumulative effect may produce rupture.

The effect of synchronous firing is about 1.4 times as great as that obtained for the same amount of powder fired in consecutive shots. Benjamin Frost says in his report on the Hoosac Tunnel that "greater depths of holes are admissible" and greater advance obtained.

FIG. 336.—Simultaneous Firing.

The sole means of obtaining perfect simultaneity is by electricity; fuses will not burn with sufficient regularity to be relied upon. The holes are charged, the cap is inserted, two wires

take the place of fuse, and the tamping is done. The wires are separated, each one being connected with its neighbor in such manner (Fig. 336) that a continuous-wire circuit *E* extends throughout the series of holes. *A* is the fuse, *B* the cartridge, and *C* the cap.

The Electric Fuse.—The several holes to be fired simultaneously are charged with an electric fuse, *A* (Fig. 337), filled with fulminates, *B*; the latter is detonated by the incandescence of a fine platinum thread, *E*, from which two copper wires, *C*, lead. The fuse-wires are of standard lengths, usually 4 or 6 feet for hand-driven holes and 16 or 18 feet for machine-drilled holes.

The ends of the fuse-wires are cleaned and twisted on the connecting wires *D* (Fig. 336), and when all these have been connected, the circuit is completed by the final connections with the machine. The leading wire *E* is usually wound on reels to pay out a supply as needed for the work. The connecting wires are of common copper wire and no larger in diameter than is necessary to transmit the current freely. By a judicious division of the mine into districts the amount of wiring may be materially reduced.

FIG. 337.—The Electric Fuse.

In Fig. 339 is shown a section of the magneto machine, which consists of a weak principal magnet, *A*, with an armature, *B*, revolving between its poles; said armature is rotated by the rack-and-pinion, *C*, operated from the handle on the down stroke. As the rack-bar strikes the spring, *D*, at the bottom, it breaks the continuous current between the two platinum bearings, *E*, and causes it to pass through the outside circuit, the fuses, etc. A machine which will fire 16 shots at one time costs $25; one good for 70 can be bought for $55.

Much wire is buried in the débris of shooting. In a mine

firing regularly 80 shots at a time, the loss from this source is 2 lbs. per day; in a small double-track mine tunnel, 2 ounces of connecting wire and 7 exploders per lineal foot; and in a large railroad tunnel, 0.32 feet of connecting wire, 0.07 feet leading wire, and 0.7 exploder per cubic yard of rock removed.

FIG. 338. FIG. 339.

A Magneto Machine and its Mechanism.

When the fuses have been connected, the last man to retire raises the handle (Fig. 338) and with one plunge ignites all the shots. An increased efficiency of the explosive is not the sole advantage of this method of firing. It eliminates the numerous accidents due to premature explosions or to delayed shots. There is but one electric current and one igniting source. Failing to fire them, there can be no ignition later. It saves the time otherwise wasted by the men, who with their neighbors must retire periodically to shelter. It permits prompt firing from a

FIG. 340.—The Cyclone Powder Loader.

safe distance. It is equally adapted to black powder and to dynamite and to flameless powders.

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GLOSSARY OF MINING TERMS.

Absolute pressure. The pressure reckoned from a vacuum.

Absolute temperature. The temperature reckoned from $-461°$ F. or $-273°$ C.

Adit. A horizontal passage from daylight into a mine, and *on* or *along* the vein. It differs from a *tunnel* in that it is not through country rock across *to* the vein; it differs from a *drift*, which has neither end in daylight.

Aerophone. A respirator in the form of a tank, receiving the exhalations from the lungs, which contains chemicals designed to revive the air and render it fit for breathing. It is used by rescuers after mine accidents.

After-damp. An irrespirable gas remaining after an explosion of fire-damp.

Anemometer. An appliance for measuring the velocity of an air-current.

Anticlinal. A fold of the rock or strata, convex upward.

Apex. The edge of a vein nearest to the surface.

Back pressure. The loss, expressed in pounds per square inch, due to getting the steam out of the cylinder after it has done its work.

Balance-bob. A heavy triangular truss, the long horizontal arm of which supports a weight at one end ballasted to balance the weight of pump-rods at the other.

Barrier pillars. Pillars, larger than the ordinary, left at stated intervals to prevent the crushing of the roof from extending beyond the section enclosed by them.

Bar-timbering. A system of supporting a tunnel roof by

long top bars while the whole lower tunnel-core is taken out, leaving an open space for the masons to run up the arching. Under certain conditions the bars are withdrawn after the masonry is completed, otherwise they are bricked in and not drawn.

Battery. The lower platform of a coal-chute. A term also used to define a timber bulkhead in a gallery.

Bearing-up stop. A partition of brattice or plank that serves to conduct air to a face.

Bed. A seam of mineral occurring among the stratified rock.

Bench. The divisions of a coal-bed caused by seams of clay or slate; also used to express the artificial divisions in the process of mining; also a terrace at the outcrop of a seam.

Black-damp. Carbonic acid gas.

Blast. The operation of forcing air by blowing. The operation of exploding powder or other agents.

Blind drift. A horizontal passage in the mine not yet connected with the other workings.

Blind lead or *blind lode.* A vein having no outcrop.

Block coal. Coal that breaks freely into rectangular blocks.

Blossom rock. The rock detached from a vein, but which has not been transported.

Blower. A discharge of gas from coal; also a fan for forcing air into a mine.

Blow-out. The decomposed mineral exposure of a vein.

Bluff. Blunt.

Bob. See *Balance-bob.*

Brattice. A canvas or plank partition, nailed to posts longitudinally, with a level or shaft to divide the same into two compartments for the purposes of separating two air-currents.

Break-through. A narrow passage cut through a pillar connecting rooms.

Breast. The face of a gallery or heading.

Buggy. A small mine-wagon for conveying coal from face to gangway.

Bulling-bar. An iron bar used to pound clay into the crevices crossing a bore-hole, which is thus rendered gas-tight.

Bull-pump. A single-acting direct pump consisting of a steam-cylinder placed over the shaft. The steam drives its piston, to which the pump-rods are attached. By means of the piston attached to the rods the water is lifted by the steam pressure. The down stroke is effected by the weight of the pump-rods.

Buntons. Timbers placed horizontally across a shaft. They serve to brace the wall-plates of the shaft-lining, and also, by means of planks nailed to them, to form separate compartments for hoisting or ladder ways.

Butt. The end faces of coal.

Butt-entry. The gallery driven at right angles with the butt-joint.

Butty. A miner working on contract.

Cage. An elevator platform.

Cap. A term signifying the point at which a vein is contracted; it is also the rock covering the ore.

Cartridge. A paper tube filled with an explosive.

Casing. The tubing of a well-hole to prevent caving.

Centre. A temporary support, serving at the same time as a guide to the masons, placed under an arch during the progress of its construction.

Chain pillar. An untouched block of mineral on either side of the gangway.

Choke-damp. Carbonic acid gas.

Chute. An inclined trough or timbered shaft through which ore is delivered by gravity to a receptacle below. See also *Shoot.*

Clack. A pump-valve.

Clearance. The space between the piston at the end of its stroke and the valve face, or the end of the cylinder.

Cleat. A joint produced by the natural tendency of coal or rock to cleave or split in a certain direction not parallel to the plane of bedding.

Coal-measures. The strata embracing the carboniferous coals.

Column-pipe. The line of pipe through which the mine-water is pumped.

Compression. In steam practice, the action of the piston in

compressing the steam remaining in the cylinder, after the closure of exhaust-valves, into the clearance space.

Contact vein. A vein lying between two dissimilar rock masses or strata.

Counter. A cross vein.

Counterbalance; Counterpoise. A weight used to balance another weight or the vibrating parts of machinery.

Counter gangway. One which is driven diagonally to the rise until the workings are reached, when it turns off parallel to the main haulageway.

Country rock. The main rock of the region through which the veins cut, or that surrounding the veins.

Crab. An iron windlass for moving heavy weights.

Creep. The crushing of the overlying rock resulting in the floor rising.

Crib. A framework built like a log-cabin. It may be a mere pillar afterwards filled with rock, or it may be the lining to a mill-hole or shaft.

Cross-course. An intersecting vein.

Cross-cut. A horizontal passage driven across the country rock to a vein.

Cross-heading. A transverse drift which is driven for purposes of ventilation from one gangway to another.

Culm. The fine waste of coal-mines containing dirt as well as coal-dust.

Curb. A timber frame intended as a support or foundation for the lining of a shaft.

Cutter. A term employed in speaking of any coal- or rock-cutting machines, the men operating them, or the men engaged in underholing by pick or drill.

Dam. A bulkhead.

Day-shift. The gang of miners working during the day-time.

Dead. The valueless matter of a vein, also gangue, or waste. It is usually made the stowing or filling of an excavated portion of the seam, bed, or vein, and then is called gob, waste, or

stull dirt. Sometimes the term is employed in speaking of a sluggish ventilating current.

Dead quartz. Quartz carrying no mineral.

Dead work. Exploratory or prospecting work that is not directly productive.

Deposit. Irregular ore bodies—not veins.

Dip. The angle, measured by the steepest line in the plane of a layer of rock, from the horizon.

Dirt-fault. A partial replacement of coal in a seam by clay. Not a true fault.

Drag. The point of union of two veins which meet without intersecting.

Drift. Any subterranean horizontal passage. See *Adit.*

Driving. Excavating drifts, adits, or levels.

Drum. The cylinder on which a hoisting-cable is wound.

Dumb drift. A gallery which conducts the air around a ventilating furnace to the up-cast shaft.

Dump. The pile of rock which has been hoisted to the surface and deposited there. It may be said to be a low-grade ore reserve.

Duty. The unit of measure of the work of a pumping-engine expressed in foot-pounds of work obtained from a bushel or 100 lbs. of fuel.

Dyke. A fissure filled with igneous matter.

Empty or *empties.* An unloaded car or the track along which it travels.

Exploder. A chemical employed for the instantaneous explosion of powder.

Exploitation. The working of a mine and similar undertakings; the examination instituted for that purpose.

Eye. The hole in a pick- or hammer-head for receiving the handle.

Face. The exposure of rock at which work is being performed.

Fathom. A volume of rock equal to six feet square multiplied by the thickness of the vein.

Fault. The dislocation of a vein. The term is also improperly used in coal-seams. See also *Rock-fault* or *Dirt-fault.*

Fire-damp. A carburetted hydrogen gas, inflammable and specifically lighter than air.

Fire-setting. The process of exposing very hard rock to intense heat, rendering it thereby easier of breaking down.

Fissure vein. Any mineralized crevice in the rock of very great depth.

Float. Broken and transported particles or bowlders of vein matter.

Floor. The stratum below a mineral bed.

Foot-wall. The face of rock below the vein.

Forepoling. A mode of timbering.

Free. A term employed in speaking of loose mineral.

Fuse. A tube, ribbon, or wire filled or saturated with a combustible compound, used for exploding powder.

Gad. An iron or steel wedge; a chisel-bit pick.

Gallery. A horizontal passage.

Gallows-frame. The frame supporting a pulley, over which the hoisting-rope passes to the engine.

Gangue. The barren portion of the vein.

Gangway. The principal level of a coal-mine.

Gash-vein. A mineralized fissure that extends only a short distance vertically. It may be confined to a single stratum of rock, but is a comparatively shallow vein.

Gateway. A gangway having ventilating doors.

Gauge pressure. The pressure shown by an ordinary steam-gauge. It is the absolute pressure plus that of the atmosphere.

Goaf. The excavated space of a coal-mine, usually filled with the valueless portion (*gob*) of the seam.

Gobbing-up. Filling with waste.

Gouge. The layer of clay or decomposed rock which lies along the wall or walls of a vein. It is not always valueless.

Guide. The timbers nailed to the timbers of a shaft for the purpose of guiding the cage.

Gunboat. A skip; a self-dumping box used in slopes.

Hanging-wall. The wall of rock above the vein.

Head-gear. A derrick.

Heading. A drift or airway. The section of tunnel driven in advance of the lower section or bench.

Heave. A dislocation of the strata.

Helve. A handle.

Hewer. A coal-miner.

Hitch. A dislocation of a vein. Also, a shoulder or hollow cut in the rock to support one end of a stull or other timber.

Hogback. See *Horse.*

Holing. The picking of a groove in the lower part of a coal-seam for the purpose of facilitating the breaking down of the upper mass.

Horse. A mass of country rock lying within a vein. Any irregularity cutting out a portion of the vein. See *Dirt-fault* and *Rock-fault.*

H-piece. The portion of a column pipe containing the valves of the pump.

Incline. An entry into a mine following the dip of the vein or seam.

Indicator. That which points or directs.

Some forms show the position of the cave in the shaft. Others record upon paper the pressure of the steam in an engine-cylinder at various points in the piston-stroke.

Inplace. A vein or deposit in its original position.

Intake. The entry which conducts the incoming air-current to the mine. It is synonymous with *downcast.*

Jar. That part of the drilling apparatus which takes up the shock of impact of the falling tools upon the bottom of the hole.

Kibble. An iron ore-bucket.

Kirving. See *Holing.*

Lagging. The slabs or small timber placed between the main timber sets and the roof or walls to prevent small rock from falling into the drift.

Lath. A plank laid over a framed centre or used in poling.

Laundry-box. The box at the surface receiving the water pumped up from below.

Level. A horizontal passage in a vein-mine, numbered 1, 2, 3, etc., consecutively from the surface, or from the cross-cut tunnel, down.

Lift. All the mine workings connected with, opened from, and mined out at one level; also the length of pump-pipe between stations.

Live quartz. A variety of quartz usually associated with mineral.

Location. A mining claim.

Lode. A mineralized fissure.

Longwall. The system of mining coal without leaving any pillars.

Man-engine. A mechanical appliance for raising and lowering miners.

Manhole. A hole or an auxiliary shaft through which a man may pass in going from one level to another, into a stope, or from one ladder to another.

Measures. A term embracing the strata of a geological series.

Mill-hole. An auxiliary shaft connecting a stope or other excavation with the level below.

Mill-run. A test of the value of a quantity of ore as distinguished from an *assay,* which tests "pocket specimens."

Mineral. Any constituent of the earth's crust that has a definite composition.

Mineral oil. Petroleum or other liquid obtained from the earth.

Miner's inch. The unit of measurement of water used by the sluice-miners. It is that amount of water hourly discharged through an opening 1 inch square under a head of several inches. If the head is 7 inches and the hole is through a plank 2 inches thick, a miner's inch is equal to about 90 cubic feet per hour.

Mining. In its broad sense embraces all that is concerned with the production of minerals and their complete utilization.

Mining retreating. A process of mining by which the vein is untouched until after all the gangways, etc., are driven, when the mineral extraction begins at the boundary and progresses toward the shaft.

Moil. A short length of steel rod tapered to a point, used for cutting hitches, etc.

Narrow work. Working places narrower than the rooms, entries, headings, break-throughs, gangways, etc.

Nitro. A corrupted abbreviation for nitroglycerine or dynamite.

Ore. A mineral of sufficient value (as to quality and quantity) which may be mined with profit.

Ore-shoot. A large and usually rich aggregation of mineral in a vein. Distinguished from *pay-streak* in that it is a more or less vertical zone or chimney of rich vein matter extending from wall to wall and having a definite width laterally.

Outcrop. The exposed portion of a vein on the surface.

Outlet. An exit passage from the mine.

Output. The product of a mine.

Panel. The division of a mine which is isolated from neighboring districts and provided with distinctive haulage and mining systems.

Parting. A joint in the rock, or a crevice in a seam, filled with clay or slate; a switch or turnout to allow loaded and empty cars to pass one another.

Pay-streak. The thin layer of a vein which contains the pay-ore.

Pike. A pick.

Pinch. A contraction in the vein.

Pipe. An elongated body of mineral. Also the name given to the fossil trunks of trees found in coal-veins.

Pit. A shaft.

Pitch. See *Dip.*

Pit-man. The shaft-man who attends to the shaft equipments, pumps, etc.

Placer. A surface accumulation of mineral in the wash of streams.

Plane. An inclined tramway for lowering cars by gravity or raising them by means of a stationary engine.

Plat. A platform. A swinging or revolving door used intermittently to connect two trackways.

Plumb (adjective). Vertical.

Plummet. A string or fine copper wire attached to a heavy weight; used for determining the verticality of shaft-timbering.

Plunger. The solid piston of a force-pump.

Pocket. A rich and large body of ore in the vein.

Poling. The process of timbering by the use of poles, for timbering in soft ground.

Poll-pick. A combination pick- and hammer-head.

Poppet; also *puppet.* A pulley-frame or the head-gear over a shaft. A valve which lifts bodily from its seat instead of being hinged.

Post and stall. See *Pillar and Room.*

Power-drill. A rock-drill employing steam, air, or electricity as a motor.

Prerelease. The act of discharging steam or air from the cylinder before the piston has reached the end of its stroke.

Prop. A piece of timber or metal placed normally to the roof or wall for its support.

Prospect. The name given to underground workings the value of which has not yet been made manifest. A prospect is to a mine what mineral is to ore.

Prospecting. The process of seeking pay-ore or the preliminary operations of a mine.

Pump. Any mechanism for raising water out of a mine.

Quick (adjective). Soft, running ground; an ore or pay-streak is said to be quickening when the associated minerals indicate richer mineral ahead.

Rafter timbering. That in which the timbers appear like roof-rafters.

Reacher. A slim prop reaching from one wall to the other.

Reamer. An enlarging tool.

Reef. The outcrop of a hard vein projecting above the surface. Also applied to auriferous quartz lodes.

Regulator. A sliding-door to apportion the amount of air to be admitted into a section of the mine.

Rib. A pillar of vein-matter left to support roof or wall.

Robbing. The taking of mineral from pillars.

Rock-fault. A disturbed portion of a vein in which coal is replaced by sandstone.

Roof. The stratum overhead.

Room. A working place in a flat mine; corresponds to *stope* in a steep vein.

Run. A mode of contract work in which steep parts of coal-seams are driven and paid for by the lineal foot or yard of progress.

Saddle. The ridge of a stratum or ore-bed.

Safety-cage. One supplied with safety appliances.

Safety-lamp. A lamp in which the flame is protected from immediate contact with the surrounding atmosphere.

Salting. Placing foreign ore in the crevices of a vein or elsewhere to fraudulently raise its apparent value.

Samson-post. An upright supporting the working-beam which communicates oscillatory motion to pump or drill-rod.

Sand-pump. See *Sludger.*

Scale. The incrustation deposited in boilers from evaporated waters.

Scraper. A tool for cleaning out drill-holes.

Seam. A layer of mineral.

Seed-bag. A water-tight packing of flaxseed around the tube of a drill-hole to prevent the influx into the hole of water from above.

Selvage. See *Gouge.*

Sett or *Set.* A frame of timber.

Shaft. A vertical opening from the surface.

Shaly. Brittle ground.

Sheave. A grooved wheel over which a rope is turned.

Shell-pump. See *Sludger.*

Shelly. Broken ground.

Shift. The duration of day's work—from six to ten hours.

Shoot. To break rock by means of explosives.

Shute. An inclined boardway through which coal is delivered.

Sill. The floor-piece of a timber sett, or that on which the track rests.

Slack. Small dirt or coal.

Slate. Bony coal and hard clay.

Slickenside. The polished surface of the vein or its walls.

Slitter. See *Pick.*

Slope. An *incline.* It is an *inside slope* when it does not extend to the surface.

Sludger. A cylinder having an upward opening valve at the bottom, which is lowered into a bore-hole to pump out the sludge or fine rock resulting from drillings.

Smift. A slow-burning fuse.

Snore. The hole in the lower part of a sinking or Cornish pump through which water enters.

Sollar. The plank flooring of a gallery covering a gutter-way beneath. Also the platform in a shaft between two ladders.

Spears. Pump-rods.

Spilling. A process of timbering through soft ground.

Spoon. A slender iron rod with a cup-shaped projection at right angles to the rod, used for scraping drillings out of a bore-hole.

Sprag. A billet of wood used to block the wheels of a car and check its speed. Sprags are permanently used on self-acting inclines. A very steep line requires a sprag in each of the four wheels, while on a moderate pitch only one may be necessary to block a hind wheel. Also, a short prop.

Square-sett. A variety of timbering for large excavations.

Squeeze. The closing of a room by the settling of the roof or the rising of the floor. The thinning away of a seam.

Squib. A slow fuse used for igniting an explosive.

Stall. The longwall working-face, or a room.

Station. An excavation adjoining a shaft for receiving the

pump or V balance-bob, or for landing the hoisting conveyances.

Stockwerke. A mass of country rock so impregnated by a congeries of veins as that the whole must be mined together.

Stope. A step. The excavation of a vein in a series of steps.

Stoping overhand. Mining a stope upward, the flight of steps being reversed.

Stoping underhand. Mining a stope downward in such a series that presents the appearance of a flight of steps.

Stowing. The débris of a vein thrown back of a miner to support the roof or hanging wall of an excavation.

Strike. The bearing of a horizontal line through the middle of a vein.

String-rods. A line of surface rods connected rigidly for the transmission of power; used for operating small pumps in adjoining shafts from a central station.

Stull. A stick of timber or platform for supporting miners or vein-waste temporarily or permanently.

Stull dirt or *stull rock.* Material supported upon the stulls.

Stump. A pillar between the gangway and its parallel airway.

Stythe. Carbonic acid gas.

Sump. The lowest point of the workings, usually a prolongation of a shaft, into which the mine water is drained and from which it is pumped.

Swamp. A trough-shaped basin in a coal-mine.

Synclinal. The depression of a seam or stratum.

Tail rope. The secondary rope used for balance, which is attached underneath the cages of a hoisting-plant, or at the tail end of the loaded and empty trains of cars on a slope for raising the empty cars or skips.

Tamping. The process of making a bore-hole gas-tight by the use of clay.

Tempering. The act of reheating and properly cooling a bar of metal to any desired degree of elasticity.

Throughs or *Thirling*. A passage cut through a pillar to connect two rooms.

Throw. The amount of dislocation of a vein.

Tram. The pair of parallel lines of rails of a trackway.

Trammer. One who pushes cars along the track.

Trap. A door used for cutting off a ventilating current, which is occasionally opened for haulage or passage; guarded by a *trapper*.

Trend. The course of a vein.

Tribute. A system of contract mining by which the miner receives his pay out of the gross value of the ore sold, less a certain deduction for royalty to the mine-owner.

Trolley. A small carriage truck having no body. A traveller making connection between two electric wires.

Tubbing. An iron o: wooden cylindrical lining of shafts.

Tubing. The tube-lining of bore-holes.

Tunnel. A horizontal passage; properly speaking, one with both ends open to the surface, but is applied to one opening at daylight and extending across the country rock to the vein or mine.

Underhand work. Picking or drilling downward.

Underholing. See *Holing* and *Kirving*.

Upcast. An entry through which the air-current rises.

Upraise. An auxiliary shaft, a *mill-hole*, carried from one level up toward another.

Vein. A mineral deposit filling a fissure or crevice.

Vug. A cavity in the rock.

Wagon breast. One from which ore or coal can be carried by wagon.

Wall. The faces of a fissure; the sides of a gallery.

Wall-plate. The long horizontal stick in a shaft timbering frame which is parallel to the vein.

Waste. The débris of an excavation; gob; goaf.

Wedging. The material, moss or wood, used to render the shaft-lining tight.

Whim. A hoisting appliance consisting of pulley support-

ing the hoisting-rope which is wound on a drum turned by a beam attached to a horse.

Whip. A hoisting appliance consisting of a pulley supporting the hoisting-rope to which the horse is directly attached.

White-damp. The noxious gas called carbonic oxide gas.

Winch or *Windlass.* A hoisting-machine consisting of a horizontal drum operated by crank-arm and manual labor.

Winning. Recovering or mining.

Winze. A small auxiliary underground shaft sunk from an upper level.

Wire-drawing. The operation, accidental or otherwise, of reducing the pressure of steam between the boiler and the cylinder.

Working-barrel. The cylinder in which the pump-piston operates.

Workings. Any underground development from which ore is being extracted.

SIGNALLING.

The writer would suggest the following code of signalling: The engineer should signal below when he is ready to hoist by raising the bucket or cage a foot or two and lowering again. This is important, particularly for the safety of the man who may be engaged in igniting the blasts near to the place of hoist. The strokes should be made at regular intervals and a halt of a few seconds between the signals. Thus

5 bells–4 bells would mean send tools to the fourth level.

1 bell: To hoist or to stop if in motion.

2 bells: To lower.

2–1–1 bells: No more hoisting.

2–2–2 bells: To change buckets from ore to water.

3 bells: Man on board; lower or hoist slow.

4 bells: Stop or start the pump.

2–2 bells: Stop or start the air-compressor.

5 bells: Send tools down.

6 bells: Send timbers down.

USEFUL INFORMATION.

The area of a circle is 0.7854 $(\text{diameter})^2$.

Ratio of area to circumference is as its radius is to 2.

An acre is 43,560 square feet.

A ton contains 2000 lbs. or 29,166$\frac{2}{3}$ troy ounces.

A troy pound = 0.822857 avoirdupois pound.

A troy ounce = 437.5 grains.

An avoirdupois pound = 7000 grains.

A troy pound = 5760 grains.

A cubic foot of gold = $300,000.

A cubic foot of silver = $10,000.

A long ton is 2240 lbs.; a short ton 2000 lbs.

A bushel of coke = 40 lbs.

A bushel of bituminous coal = 76 lbs. = 2688 cu. in.

1000 feet (board measure) of dry white pine = 4000 lbs.

1000 feet (board measure) of green white pine = 6000 lbs.

One cord of seasoned wood = 128 cu. ft.

A mile of track (rails 16 lbs. per yd.) weighs 25 tons 320 lbs.

A mile of track (rails 25 lbs. per yd.) weighs 39 tons 640 lbs.

A mile of track (rails 35 lbs. per yd.) weighs 55 tons 0 lbs.

A mile of track requires 9 kegs (1780 lbs.) of 3-in. spikes.

A mile of track requires 15 kegs (3110 lbs.) of 4½-in. spikes.

A mile of track requires 20 kegs (3960 lbs.) of 4½-in. spikes.

A mile of track requires 2640 cross-ties (2 ft. apart).

A mile of track requires 528 splice-joints (2 bars, 4 bolts and nuts per joint), each weighing 5 to 10 lbs.

One miner's inch = 2159 cu. ft. per 24 hours = 0.025 cu. ft. per sec.

The pressure of 1 atmosphere is represented by a column of mercury at 32° F., of a height equal to 0.760 metre = 29.92 inches = 14.701 lbs. per sq. in., or by a column of water 33.94 ft. high.

1 lb. air at	32° F.,	14.7 lbs.	pressure,	occupies		12.387	cu. ft.
1 " "	62° F.,	"	"	"	"	13.141	"
1 " "	180° F.,	"	"	"	"	16.106	"
1 " "	212° F.,	"	"	"	"	16.910	"
1 " water at	62° F.,	"	"	"	"	0.016	"

1 cu. ft. water at	39° F. = 7.49 gallons,	weighs	62.425	lbs.
1 " " "	62° F. = " "	"	62.355	"
1 gallon " "	62° F. = 231 cu. in.,	"	8.325	"
1 cu. in. " "	62° F. = 0.00434 gal.,	"	0.036	"
1 ton " "	62° F. = 240.3 gals.,	occupies	35.9	cu. ft.

A col. of water,	1 sq. in. base,	33.947 ft. high	= 1 atmos.	
" " " "	1 " "	27.7 in. high	= 1 lb. press.	
" " " "	1 " "	1 ft. high	= 0.434	lb. pr.
" " " mercury,	1 " "	1 in. "	= 0.4914	"
A bar wro't iron,	1 " area,	1 ft. long	= 3.33	lbs. wt.
" " " "	1 in. diam.,	1 " "	= 2.618	"

TABLE OF WEIGHTS OF VARIOUS SUBSTANCES.

	Weight per Cu. Ft.	Cu. Ft. per Long Ton.		Weight per Cu. Ft.	Cu. Ft. per Long Ton.
Gold	1203	1.87	Syenite	147 to 184	15.2 to 12.1
Lead	710	3.15	Porphyry	166 to 171	13.5 to 13.1
Silver	625	3.42	Slate	162 to 178	13.8 to 12.6
Rolled iron	480	4.68	Quartz	165.2	13.6
Galena	468	4.82	Sandstone	130 to 157	17.3 to 14.3
Nickel glance	468	4.82	Brick	125 to 135	18.1 to 16
Cerusite	400	5.60	Clay	119.7	18.7
Chalcocite	355.7	6.30	Anthracite	85.4 to 99	26.2 to 22.6
Magnetite	338.6	6.63	Bituminous	75 to 83	29.8 to 26.1
Specular iron ore	327.4	6.84	Cannel	75	29.8
Pyrites	312	7.05	Lignite	78 to 84	28.7 to 27
Barytes	277.5	8.07	Oak	73	30.6
Chalcopyrite	262.1	8.55	Ash	52.4	42.7
Zinc blende	250	8.96	White pine	44.3	50.5
Hematite	250	8.96	Yellow pine	38.7	57.8
Limestone	168	13.3	Wood charcoal	25 to 39	89.6 to 57.4

EQUIVALENTS OF FRENCH AND ENGLISH MEASURES.

ABBREVIATIONS. — M. = metre; cm. = centimetre; G. = gramme; L. = litre; ft. = foot; lb. = pound; in. = inch; oz. = ounce; dwt. = pennyweight; gr. = grain; yd. = yard; gal. = gallon; T. = troy; A. = avoirdupois; sec. = second; sq. = square; cu. = cubic; h.-u. = heat-unit.

1 M.	= 3.28 ft.
1 M.	= 39.39 in.
1 ft.	= 0.3048 M.
1 in.	= 0.0254 M.
1 yd.	= 0.9144 M.
1 Gunter's chain	= 20.1168 M.
1 mile	= 1609.35 M.
1 sq. M.	= 1.2 sq. yd.
1 sq. yd.	= 0.836 sq. M.
1 sq. in.	= 0.00065 sq. M.
1 sq. M.	= 1555.2 sq. in.
1 acre	= 4048 sq. M.
1 cu. in.	= 0.0000164 cu. M.
1 cu. ft.	= 0.02832 cu. M.
1 cu. M.	= 1.31 cu. yd.
1 G.	= 15.43 gr.
1 G.	= 0.0022 lb. A.
1 T. gr.	= 0.0648 G.
1 T. lb.	= 5760 gr.
1 T. lb.	= 373.242 G.
1 A. lb.	= 453.593 G.
1000 G.	= 2 lb., 8 oz., 3 dwt., 0.35 gr. T.
1000 G.	= 2 lb., 3 oz., 4 dr., 10.473 gr. A.
1 fluid oz.	= 0.02957 L.
1 quart	= 0.9464 L.
1 gal.	= 3.78543 L.
1 atmosphere	= 0.760 M.
1 ft. per sec.	= 0.305 M. per sec.
1 mile per hour	= 0.447 M. per sec.
1 M. per sec.	= 3.281 ft. per sec.
1000 M. per hour	= 0.621 mile per hour.
1000 G. per sq. M.	= 0.205 lb. per sq. ft.
1000 G. per cu. M.	= 0.0624 lb. per cu. ft.
1 lb. per sq. ft.	= 4883 G. per sq. M.
1 lb. per sq. in.	= 703077.0 G. per sq. M.
1 ton per ft.	= 3,333,333 G. per M.
1 gal. per sq. ft.	= 48.905 L. per sq. M.
1 L. per sq. M.	= 0.0204 gal. per sq. ft.
1 G. per L.	= 70.116 gr. per gal.
1 lb. per cu. ft.	= 16020 G. per cu. M.
1 cu. ft. per lb.	= 0.0624 cu. M. per 1000 G.
1 degree Fahr.	= 0.5555 deg. centigrade.
1 degree Cent.	= 1.8 deg. Fahrenheit.
1 lb. per sq. ft.	= column of mercury 0.00359 M. high.
1 L. of normal air	= 19.955 grains.
1 G. M.	= 0.007233 ft.-lb.
1 ft.-lb.	= 138.2 G. M.
772 ft.-lbs.	= 106700 G. M.
1 calorie	= 3.968 heat-units.
1 heat-unit	= 0.252 calorie.
1 thermal unit	= 0.4536 calorie.
1 h.-u. per lb.	= 0.5555 calorie per 1000 G.
1 calorie per 1000 G.	= 1.8 h.-u. per lb.

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